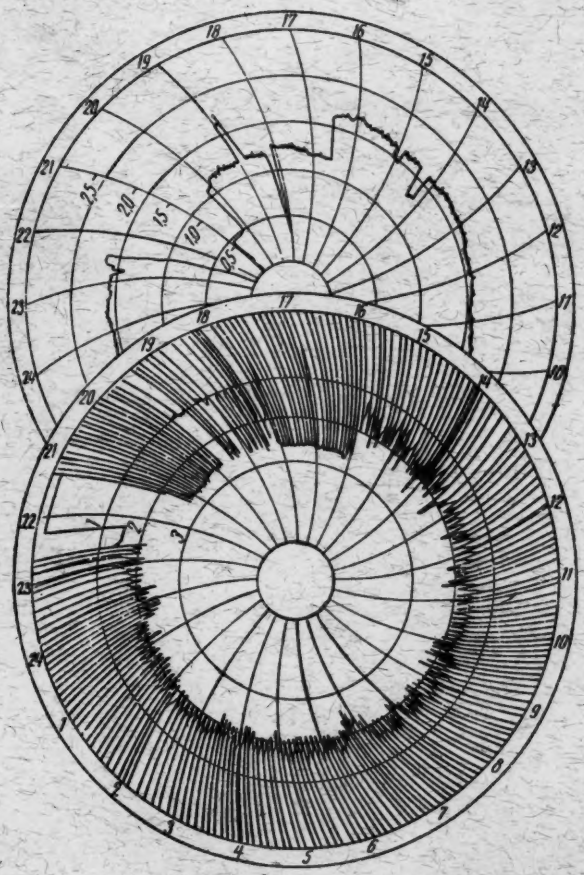
Number 12

December 1958

METALLURGIST

translated from Russian



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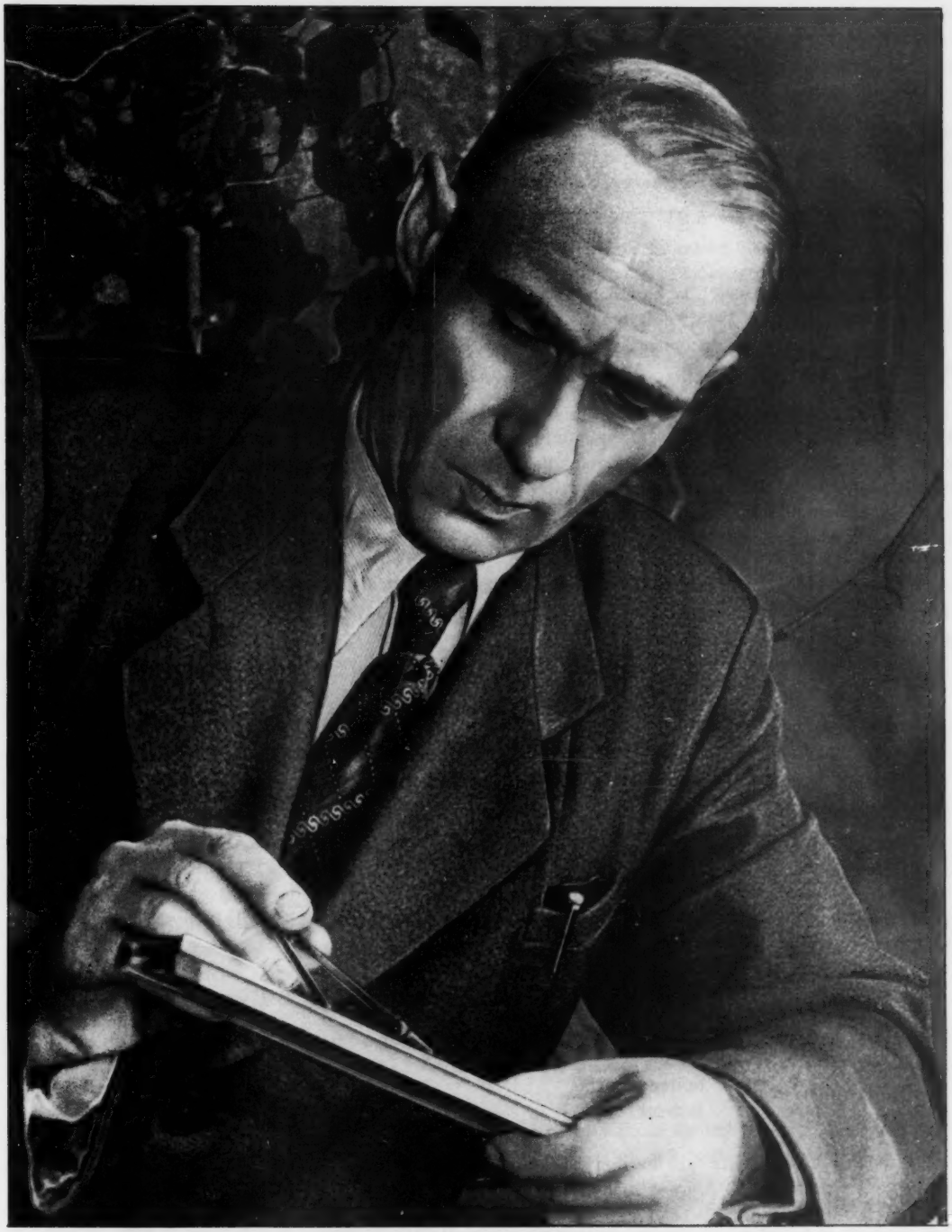

The senior foreman of the heavy-section mill, Boris Iakovlevich Grigor'ev has been engaged on modernization at the Serov Metallurgical Combine since 1951. During this time he has introduced a number of effective innovations which have resulted in a saving of state resources amounting to one million rubles annually.

Photograph above: One of the best innovators of the A. K. Servo Metallurgical Combine, Boris Iakovlevich Grigor'ev.

Photo A. Gireva

THE IRON AND STEEL INDUSTRY OF THE USSR IN 1959-1965

The theses of Comrade N. S. Khrushchev's speech "Planned figures of the development of national economy of the USSR for 1959-1965," delivered at the XXIst Congress of the Communist Party of the Soviet Union, are a very important historic document. They show how our country advances with rapid strides on the road of communism.

In his speech delivered on November 14, 1958, at the reception of the graduates of military academies, Comrade N. S. Khrushchev said that the main object of the seven-year plan, which is a constituent part of the program for the next 15 years for building communism in our country,"is to ensure a further extensive development of all branches of the national economy based on a preferential expansion of heavy industry and a strengthening of the economic potential of our country, and on this basis, to bring about a new substantial rise in the living standards of the people."

Special attention is paid to the need for increasing ferrous metal production substantially.

The economic plan for 1959-1965 envisages the following output of the main branches of the iron and steel industry, in million tons:

	1958 expected output	1965 planned figures	1965 in % compared with 1958
Pig iron	39.5	65-70	165-177
Steel	55.0	86-91	155-165
Rolled products	42.7	65-70	152-164
Iron ore	88.2	150-160	170-181

The average annual increase in the output of the above goods in 1959-1965 will be greater than in the preceding seven-year period as shown below, in million tons:

	1952-1958	1959-1965
Pig iron	2.5	3.6-4.4
Steel	3.4	4.4-5.1
Rolled products	2.7	3.2-3.9
Iron ore	6.2	9-10.3

It is seen from the figures that the iron and steel industry and, in particular, iron ore production will be developing at a rapid rate.

In addition to a substantial increase in iron ore output at existing mines, it is planned to start mining operations at new iron ore fields. The iron ore output will be expanded mainly by open-pit mining. Plans have been drawn for the construction of large ore-beneficiating combines which will permit a thorough beneficiation of iron ores, an increase in the iron content in commercial ores and, hence, an increase in the output of blast furnaces.

The output of alloy and low-alloy steels will be increased substantially and this, together with the development of the thermal treatment of steel, will improve the quality of rolled product.

Electric steel production will increase by a factor of 1.7-2, the production of plate steel by a factor of about 2. The number and the sizes of sections and tubes rolled will be extended. The most rapid expansion will take place in the output of economical light (reduced) sections and periodic sections; the mass production of bent sections from steel plate and strip is under way.

New units will be constructed and put into operation to increase the productive capacities for pig iron by 24-30 million tons compared with a 16.1 million ton increase in 1952-1958; for steel, by 28-36 million tons compared with a 12.4 million ton increase in 1952-1958; for rolled product, by 23-29 million tons compared with a 6.9 million ton increase in 1952-1958.

The technical level of the iron and steel industry of the USSR will be raised substantially. Large metallurgical mechanized and automatic units fitted with modern equipment will be constructed: large blast furnaces of 1719-2000 or more cu m useful volume, large capacity open-hearth furnaces (250 and 500 tons), electric steel-melting furnaces of 80 and 180 ton capacity, and 65-75 ton converters. Highly efficient rolling mills, tube mills and finishing machines which will ensure continuous production processes, quality control of the product, complex mechanization and automation of industrial operations, will be set up.

In 1965, the Ukrainian SSR will overtake the most developed capitalist countries' output per capita in several basic industries. Thus, in 1965, the output of pig iron per capita in the Ukrainian SSR will be 1.7 as big as the output in the USA in 1957, 1.9 as big as in West Germany, 2.5 as big as in France and England; the output of steel will be 1.2 of the 1957 steel output of the USA, 1.4 that of West Germany, 1.6 that of England and 2.2 that of France.

Capital investment in the iron and steel industry for the seven-year period is to be about 100 billion rubles. This 2.4 times as much as capital investments in the iron and steel industry for the 1952-1958 period. Capital investments in the development of the iron-ore industry will increase by a factor of 2.8. The money allotted will be concentrated on the minimum possible number of projects. About 60% of the total capital outlay in the iron and steel industry will be earmarked for the building of 15 very important iron and steel works and 15 ore-mining establishments.

As regards the distribution of productive capacities, the coming seven-year plan envisages changes in the direction of speeded-up development of the country's eastern regions. Their contribution to the All-Union industrial production will increase and in 1965 it will constitute approximately 43% in the pig-iron output, 47% in the steel and 48% in the rolled products.

The pig iron output in the RSFSR will increase by 70-85%, the steel output by 60-69%. Of great importance is the envisaged work on the development of the iron ore deposits of the Kursk Magnetic Anomaly. The Cherepovets Works will be completed and the Novolipetsk Metallurgical Works will be greatly expanded. The Ural regions will retain the first place in the production of ferrous metals in the RSFSR. The productive capacity of the Magnitogorsk, the Orsk-Khalilovsk and Nizhne-Tagil Metallurgical Combines, the Cheliabinsk Metallurgical Works, the Sinarsk and the Cheliabinsk Tube Works will increase substantially, and the first stage in the construction of the Kachkanar Ore-Beneficiation Combine will be completed.

The utilization of the enormous natural resources of Siberia will proceed on a large scale. It is planned to build two large iron and steel works there, which will be the basis of the third iron and steel center of the USSR.

In the iron and steel industry of the Ukrainian SSR the plan includes the task of ensuring a predominant development of the iron-ore and manganese industries and the erection of 5 large ore-beneficiation combines in the Krivoi Rog iron-ore basin and an ore-beneficiation combine in the newly developed Kremenchug iron ore district. The Kamysh-Burun Combine will be enlarged. These projects will make it possible to increase the iron ore output of the Ukrainian SSR by a factor of 1.6 in the next 7 years.

In the next seven-year period, the pig iron output in the Ukrainian SSR will increase by 47-57%; steel output by 38-45%; rolled product by 32-39%, and the output of the type of rolled products of which there is a shortage will increase by a factor of 2 or 3.

An extensive development of the iron and steel industry of the Kazakh SSR is envisaged. The Karaganda Metallurgical Works and the Ermakovskii Ferroalloy Works will be built. The Kazakh Republic will become an important producer of pig iron and steel in the USSR.

The iron ore industry of the Kazakh SSR will be substantially expanded by means of erecting new ore-beneficiating combines. One of the largest combines in the country, the Sokolovsko-Sarbai Ore-Beneficiation Combine, of 19 million tons iron ore annual output capacity, will be put into operation and it will provide ore for the growing iron and steel industry in the Urals and will meet the requirements of the iron and steel industry of Kazakhstan.

The iron and steel industry in other Soviet Republics is also being developed further. New manganese-ore mines and beneficiation plants are being put into operation in the Georgian SSR. The completion of the iron and steel works in the Azerbaidzhan SSR is envisaged.

In total output of ferrous metals the Soviet Union is far ahead of all European capitalist countries and is confidently catching up with the USA.

In the coming seven-year period, the iron and steel industry of the USSR will, as before, expand at a much faster rate than in the most important capitalist countries, including the USA.

The superiority of the Soviet Union in the rate of increase in ferrous metal output constitutes a practicable basis for catching up with, and overtaking, the USA in the output of ferrous metals in an unprecedentedly short time.

FULFILLING THE RESOLUTIONS OF THE XXth CONGRESS OF THE COMMUNIST PARTY OF THE SOVIET UNION

S. M. Filippov

State Planning Commission of the USSR

As a result of the successful completion of the prewar and postwar Five-Year Plans our iron and steel industry have grown into highly developed branches of the socialist economy and ensured a multiple increase in the output of pig iron, steel, rolled products, tubes and other iron and steel products. In 1955, i.e., in the last year of the fifth Five-Year Plan, our country produced 7.9 times more pig iron, 11 times more steel and 12 times more rolled products than in 1913.

As the total output increased, so assortment of the products became wider, the quality improved and the efficiency and productivity increased.

The directives of the XXth Congress of the CPSU (Communist Party of the Soviet Union) with regard to the sixth Five-Year Plan for the development of the national economy of the USSR envisaged a further substantial increase in the output of ferrous metals. Soviet metallurgists, working selflessly under the guidance of the Communist Party on fulfilling the task assigned to them, achieved great successes during the period of the sixth Five-Year Plan (Table 1).

In the three years after the XXth Congress of the CPSU, the output of pig iron in our country increased by 6.2 million tons, of steel by 9.7 million tons, of rolled products by 7.4 million tons, of steel tube by more than 1 million tons and of iron ore by 16.3 million tons. This is a remarkable achievement for the vast army of Soviet metallurgists.

The importance of the increase achieved becomes even more significant if it is compared with the increase in the productivity in three years of the biggest capitalist countries, which have a highly developed iron and steel industry (Table 2).

Thus, as far as the absolute magnitude of the increase is concerned, the expansion of the iron and steel industry of the USSR leaves all the big capitalist countries far behind.

In 1958, the iron and steel industry of the Soviet Union is expanding even more. In nine months of the

TABLE 1

Output of the Main Products of the Iron and Steel Industry of the USSR, in Thousand Tons

Product	1955	1956	1957	1958	
				expected output*	increase in 3 years
Pig iron	33 310	35 754	37 040	39 500	6 190
Steel	45 272	48 698	51 165	55 000	9 728
Rolled product	35 339	37 833	40 199	42 750	7 411
Steel tubes	3 547	3 850	4 189	4 595	1 048
Iron ore	71 862	78 079	84 169	88 400	16 538

*Estimated on the basis of 9-months production in 1958.

TABLE 2

Output of Pig Iron and Steel in Some Capitalist Countries, in Million Tons

Country	1953		1956		Increase in 3 years	
	pig iron	steel	pig iron	steel	pig iron	steel
USA	68,8	101,3	68,9	104,5	+0,1	+3,2
Fed. Ger. Rep.	11,6	15,4	17,6	23,2	+6,0	+7,8
Britain	11,4	18,0	13,4	21,0	+2,0	+3,0
France	8,7	10,0	11,5	13,4	+2,8	+3,4
Japan	4,5	7,7	6,3	11,1	+1,8	+3,4

current year the output of pig iron increased by 6% as compared with the corresponding period of last year, the output of steel and rolled products increased by 8%. In capitalist countries, the iron and steel industry is cutting down the output this year. For the period from January to July, 1958, the output of ferrous metals decreased in comparison with the corresponding period of 1957, as follows: pig iron by 37.9% and steel 34.5% in the USA; by 3.9% and 1% respectively in West Germany and by 5.9 and 6.5% respectively in Britain.

A rapid expansion of the production of ferrous metals contributed to a further increase in the share of the Soviet Union in the world output of steel from 4% in 1929 to 14.5% in 1950, 16% in 1955 and 17.7% in 1957.

The Central Committee of the CPSU and the Council of Ministers of the USSR took several steps towards an improvement in the economic and technical management of the establishments and the projects under construction and towards a reduction in working hours and a revision of wages and salaries.

The production of the iron and steel industry in the USSR increased in all the Republics which have an iron and steel industry.

The rapid development and the increase in the production was achieved by introducing new plants and by a more efficient utilization of existing units, by making use of the most recent scientific advancements in industry, and of new techniques and the experience of innovators.

The predominant part in the rise of pig iron production is played by the new plants. In 1957, the increase in pig iron output by the new blast furnace compensated for the losses due to shutting down obsolete plants and to a partial decline in the quality of raw materials at eastern iron and steel works, which decline resulted in some fall in the output of pig iron at the existing blast furnaces of some of the works.

The main contribution to the increase in steel and rolled product output was made by a more efficient utilization of existing plants. Quite a substantial part of the increase, however, is due to the introduction of new plants.

The technical progress in blast furnace production manifested itself, first of all, in an increase of the mean useful volume of blast furnaces; the mean volume of blast furnaces increased of the mean useful volume of blast furnaces; the mean volume of blast furnaces increased from 639 cu m per furnace in 1957 to 715 cu m in 1955 and 844 cu m in 1958.

TABLE 3

Basic Technical and Economic Indices of Pig Iron and Steel Production in the USSR*

Indices	1955	1958 (first half-year)
Ratio of blast furnace useful volume to daily output	0.80	0.77
Current idle periods of blast furnaces, % of nominal time of operation	0.9	0.8
Iron ore consumption, in tons per 1 ton of conversion pig iron	1.767	1.880
Sinter (included in the above figure) in tons per 1 ton of conversion pig iron	1.090	1.409
Dry coke consumptions, in tons per 1 ton of conversion pig iron	0.837	0.795
Mean daily output of steel per 1 sq m of the hearth of open-hearth furnaces, tons	6.55	7.56
Idle periods of open-hearth furnaces; % of the calendar time of operation total		
Total	14.5	9.7
Cold (included in the above figure)	7.2	4.4
Consumption of metal in tons per 1 ton of open-hearth steel	1.115	1.112
Fuel consumption in tons per 1 ton of open-hearth steel	0.203	0.185

*For specialized establishments of the National Economic Regions.

TABLE 4

Operating Statistics of the Best Blast Furnaces

Works	Ratio of blast furnace useful volume to daily output		Idle periods, % of nominal time of operation	
	1955	6 mo. 1958	1955	6 mo. 1958
Magnitogorsk	0.65	0.61	0.7	0.5
Kuznetsk	0.73	0.67	1.2	1.1
Serov	0.66	0.64	0.3	0.4
Cherepovets	0.84 (1956)	0.64	0.8 (1956)	0.4

At the same time, the design of blast furnaces was improved, the production processes were mechanized and further improvements were made in the preparation of raw materials for the blast-furnace process by means of a better blending of iron ores, the expansion of sinter production and especially by developing the production of fluxed sinter. The output of sinter increased from 33.8 million tons in 1955 to 49.2 million tons in 1958; these figures include and increase in fluxed sinter from 25.4 million tons to 47.2 million tons. The amount of fluxed sinter in the total output increased from 57% to 96%.

TABLE 5

Operating Statistics of the Best Open-Hearth Furnaces

Works	Steel output, tons per m sq per 24 hrs		Idle periods, %	
	1955	First half of 1958	1955	First half of 1958
Magnitogorsk	8.60	8.94	8.0	7.0
Kuznetsk	8.22	8.97	7.2	7.1
"Zaporozhstal"	8.26	9.03	10.9	8.5

TABLE 6

Metal Output Per Capita (kg)

Country	1913	1950	1955	1957
		Pig iron		
USSR	30	106	169	182
USA	324	391	427	420
Production in the USA was higher than in the USSR by a factor of	10.7	3.7	2.5	2.3
		Steel		
USSR	30	151	229	250
USA	327	579	642	599
Production in the USA was higher than in the USSR by a factor of	10.8	3.8	2.8	2.4

In the production of sinter and in its use, the USSR is far ahead of all the developed capitalist countries. Blast-furnace operation at elevated top pressure is extensively applied.

The number of blast furnaces operating at elevated gas pressure increased from 47 in 1955 to 65 in 1958. This advanced method has not yet been extensively introduced in the capitalist countries. In 1956, 28 blast furnaces in the USA and only 2 blast furnaces in Britain were working at high-pressure conditions.

A further introduction of operation at constant humidity and high temperature of the blast took place. This method is now employed at more than 90% of all the blast furnaces in the USSR; the production of low-manganese conversion pig iron at works in the East and the South was increased. About 90% of all the pig iron in the USSR is produced in highly mechanized and automated blast furnaces.

In steel production, technical advances manifested themselves first of all in the enlargement of steel-making plants, in the increase of their thermal capacity and the improvement of their design.

In recent years, open-hearth furnaces of 250, 370 and 500 ton capacity were built. The average capacity of newly introduced open-hearth furnaces increased from 167 tons in 1950, to 230 tons in 1955 and 271 tons in 1958; and steel output per furnace per annum increased in the same periods from 56,200 tons to 81,600 and 96,200 tons.

Our open-hearth furnaces are more efficient than open-hearth furnaces in the USA:

Furnace capacity	180-200	500
Average output per year, in thousand tons:		
USSR	175	288
USA	138	257

In the technology of steel production the most significant achievement was a further intensification of the steel-smelting process with the use of oxygen, basic refractories, speeding-up of operations and extensive mechanization of labor-consuming work. The amount of steel made with the use of oxygen increased from 4,015,000 tons to 13,860,000 tons in 1958, i.e., by a factor of 3.4. More than 90% of all open-hearth steel in the USSR is produced in furnaces with an automatic control of thermal conditions. At several iron and steel works, compressed air is fed into the ports of the open-hearth furnaces, the output being thus increased by up to 8%.

Electric-furnace steelmaking was developed in the direction of improving the technological processes, operating the existing plants more efficiently and enlarging the number of electric furnaces by introducing new large furnaces which embodied mechanical charging, basic roofs, and automatically adjustable electrodes.

A new modification of the Bessemer process was developed and adopted; by this method it is possible, by means of blowing oxygen from above into open-hearth pig iron in the basic Bessemer converter, to obtain steel of the same quality as open-hearth steel.

In rolled-steel production, a futher enlargement of the assortment and an improvement in the quality of rolled products took place: the percentage of rails and section steel in the total amount of rolled steel decreased markedly and the percentage of steel plate, wire rod and tube billets increased; the percentage of special-shape sections and economic sections as well as low-alloy steel products increased also. In 1955, 250,000 tons of low-alloy steel was rolled; in 1958, 1,385,000 tons will have been rolled i.e., 5.5 times as much.

At several works, the weight of ingots and billets was increased, heating facilities were extended, the roll design was improved, the speed of rolling was increased and the organization of repair work was improved.

Substantial improvements were introduced in the technique of rolling high-quality steel, and the technique of the production of new grades of steel and precision alloys was mastered.

Considerable technical progress was made in tube and hardware production as well as in the mining, beneficiation and sintering of ores.

A further improvement in the operation and output of basic plants, a reduction in cost and a rise in operating efficiency were achieved (Table 3), owing to continuous technical improvements in iron and steel production, the mastering and extensive adoption of new technology and the advanced techniques of leading establishments.

In the economic utilization of iron and steel plants, the Soviet ferrous industry is far ahead of the industry of capitalist countries, in particular of the USA, Britain, France and the Federal German Republic.

Several works have achieved a substantially better ratio of blast furnace useful volume to daily output and show much better results than the average figures for the USSR (Table 4).

Never resting content with what has been achieved and constantly searching for new productive potentialities, the Magnitogorsk blast-furnace operators achieve greater and greater successes. The personnel of blast furnace No. 3 attained a ratio of blast furnace useful volume to daily output equal to 0.575. The teams which operate blast furnace No. 4 have reached a ratio equal to 0.593 and reduced idle periods to 0.16%. Improvements in the efficiency of other furnaces too can be achieved.

Not all iron and steel works, however, have achieved such excellent results. As far as blast-furnace production is concerned, some works are far behind, not only with respect to the leading establishments but also with respect to the average results in the Soviet Union.

The Makeevka Works expands year by year, introduces very modern equipment and adopts mechanization and automation extensively. The technical and economic indices of its operation, however, are below those of other works. For more than three years, there has been no improvement in the ratio of blast furnace useful volume to daily output: it was 0.83 in 1955, 0.84 in 1956, 0.84 in 1957 and again 0.84 for the first half of 1958.

Is it possible to improve the efficiency of the blast furnaces at the Makeevka Works ? Of course it is, judging by the operating experience of the best units of this Works. In an article "Potential Reserves of the Makeevka Works" by a group of employers of the Makeevka Works, it is pointed out that the personnel of one blast furnace at the Makeevka Works has operated the furnace at a ratio of blast furnace useful volume to daily

output equal to 0.768 (i.e., 10% better than the average for the Works) for a long time. Almost as good efficiency is shown at another furnace of this Works. But the remaining furnaces operate much less efficiently, although the operating conditions are almost the same. At the blast furnace plant of the Makeevka Works, there are many potential sources, as yet not utilized, of output increase. Thus, the temperature of the blast is 720-730°C on the average and up to 900°C at one of the furnaces. By increasing the temperature of the blast at all the furnaces, it will be possible to obtain an increase in daily output of pig iron and a saving in coke. It is well known that good raw material is an important prerequisite for an increase in blast furnace output. The Makeevka Works must be given assistance to solve the problem of improving the quality of the ore, coke and limestone which are supplied to this Works.

By analyzing the operating statistics thoroughly, the blast-furnace operators at Makeevka will undoubtedly find new reserves, achieve a general rise and an improvement in the efficiency of the blast furnaces.

At the open-hearth plants of our iron and steel works a further improvement in the efficient use of equipment was also achieved. The average daily output of steel per 1 sq m of the hearth of the open-hearth furnaces increased from 6.55 tons in 1955 to 7.56 tons in the first half of 1958; idle periods decreased from 14.5 to 9.7%; fuel consumption per 1 ton of open-hearth steel fell from 203 to 185 kg.

Several leading works achieved even better results in open-hearth furnace operation (Table 5).

The personnel of the open-hearth plants at the Magnitogorsk and the Kuznetsk Combines, which achieved very good results in steel output in 1955, found additional sources of improvement and achieved, in the first half of 1958, 8.94 and 8.97 tons per sq m average daily output, respectively. Open-hearth furnace operators at the "Zaporozhstal" Works achieved an output of 9.03 tons per sq m and, at the same time reduced idle periods by 2.4% and increased the durability of roofs considerably.

At the same time, some works are lagging behind the leading works in the operation of open-hearth furnaces. Let us take, for instance, the Makeevka Works. Open-hearth furnace operators at this Works have had considerable successes in recent years, and have produced 7.42 tons of steel per sq m of the hearth in the first half of 1958. This, however, is 1610 kg less than at the "Zaporozhstal" Works and 1520 and 1550 kg less than at the Magnitogorsk and the Kuznetsk Combines respectively.

The open-hearth furnace operators at the Makeevka Works are lagging behind because they do not make proper use of oxygen. The application of oxygen resulted in a mere 11-12% increase in steel output, while at the "Zaporozhstal Works" the increase amounted to 20-22%.

In the analysis of the operating statistics the employees of the Makeevka Works rightly point out that the charging and the initial periods of heats at the open-hearth plant are very long. The average time of charging in the last three years was 2 hours and 25 minutes, 2 hours 50 minutes and 3 hours respectively (often it was 4-5 hours). These shortcomings are due to several causes. The problems of redesigning the charging machines for handling large-capacity charging boxes are being solved too slowly at the Works, no attention is paid to the problem of how to increase the output of the fagotting press, the steel-smelting operations are not planned efficiently and the maintenance of the furnaces is not improved.

It is seen from the figures quoted that the metallurgists of the Soviet Union have attained a high efficiency in the utilization of metallurgical equipment and are finding new potential sources of increasing their output.

Important measures carried out in the iron and steel industry of the USSR, and aimed at raising the technical level of the industry and improving the technology and the organization of production, ensured an increase in operating efficiency.

In the past two and a half years the operating efficiency of the workers in the iron and steel industry has increased by 12% in the terms of the gross output per worker.

A long time ago the Soviet Union became a country of metal industry. Having overtaken the USA in the rate of expansion and in the mean annual increase in the output of iron ore, pig iron and steel, we want, as soon as possible, to catch up with, and overtake, the USA in the output of metal per capita. The progress in the fulfillment of this task is illustrated by Table 6.

It is seen from Table 6 that the output per capita in the Soviet Union increases several times as fast as in the USA. In the period from 1913 to 1957 the output of pig iron per capita in the USSR increased by a factor of 6.1 and of steel by a factor of 8.3; the respective figures for the USA are 1.3 and 1.8. Thus, on the way to fulfilling the great task – that of catching up and overtaking America in the output of metal per capita – set out by the Communist Party, the Soviet metallurgists have made a gigantic step forward.

THE BLAST FURNACE INDUSTRY

THE ORGANIZATION AND TECHNOLOGY OF OPERATIONS ON THE RECONSTRUCTION AND REPAIRS OF BLAST FURNACES

I. D. Vikhrev

Head of the Section of Construction Organization of the Gipromez

The task of increasing the output of pig iron to 80-85 million tons in the next 15 years can be successfully fulfilled not only by building new large blast furnaces but also by the reconstruction of existing furnaces, by increasing their useful volume, installing better equipment and adopting new technological processes.

In recent years, several iron and steel works have employed an advanced method of rebuilding the furnace – the so-called method of shifting. The blast furnace which is to be rebuilt remains in operation until all the work on the new furnace, which is assembled on a special platform some distance away, has been finished. The old furnace is then shut and pulled down, and the new one is moved into the design position and placed on the permanent foundation.

When this method of shifting is employed,the time of reconstruction is considerably reduced (40-45 days instead of 80-90 days) and the working and safety conditions are improved because most of the work is carried out under normal conditions without a large concentration of workers which carry out normal range of operations.

For shifting the furnace, not more than four guide tracks should be laid, and for shifting the furnace without lining two guide tracks may be adequate. The furnace should be shifted at first by means of hydraulic jacks and then with hoists.

All steel constructions, castings and equipment, including the charging apparatus, should be assembled and all work with refractory materials should be completed on the temporary platform.

If the furnace to be shifted includes refractory brickwork, the reconstruction of a blast furnace built according to standard design can be reduced to 30 days. In this case, the assembly work and the brick laying can be so planned as to have as few operations as possible carried out simultaneously without concentrating a large number of workers.

There are three designs for the platform for furnace shifting: a platform can be constructed of reinforced-concrete blocks (Fig. 1), of reinforced-concrete tubes (Fig. 2) or of steel (Fig. 3).

The construction of the platform with reinforced-concrete sectional blocks, which can be used for industrial buildings or dwelling houses after the reconstruction of the furnace, should be regarded as the most economical proposition. In addition, when the sectional blocks are employed, the amount of steel used is reduced and the construction work is simplified.

The reinforced-concrete sectional blocks are laid on a concrete bed, 150-200 mm thick, and on top of the block is placed a one-piece reinforced-concrete plate with projecting reinforcement bars to which are welded the H-beams which serve as the guide rails for the furnace. The bogies – rollers joined by planks – on which railroad rails are laid, can be moved on these tracks. The bogies support the hearth bottom which consists of H-beams enclosed between the top and bottom plate flooring. The spaces between the H-beams can be utilized as channels for the air-cooling of the hearth bottom.

If, owing to the shortage of space, it is not possible to assemble the furnace on one platform, the assembly work can be carried out on two platforms set up above the pig-iron and slag disposal track. On one side of the

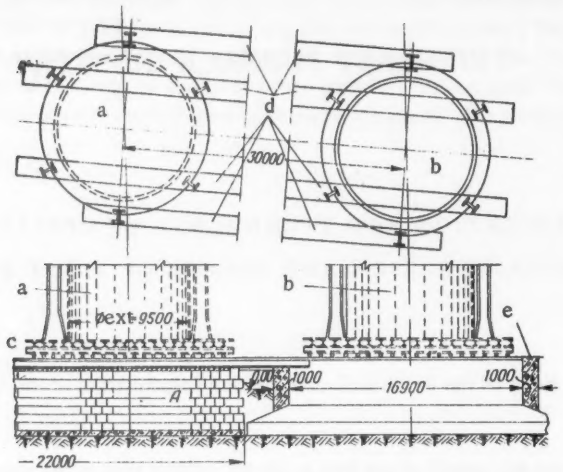

Fig. 1. Platform made of reinforced-concrete blocks (A) for shifting a lined blast furnace
a) furnace on the platform; b) furnace on the foundation; c) bogies for shifting the furnace; d) guide tracks; e) reinforced part of the existing foundation.

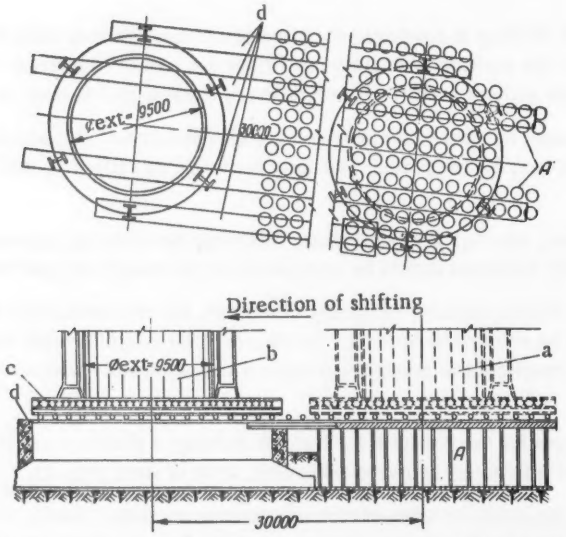

Fig. 2. Platform made of reinforced-concrete tubes (A) for assembling and shifting a lined blast furnace (notation as in Fig. 1).

platform the hearth of the furnace is assembled and on the other, the upper part of the furnace beginning with the tuyere section or the mantle ring and including the shaft with the lining and the furnace top.

The method of shifting can be used for the reconstruction of machine buildings, skip pits, blast-furnace stoves, stockhouses and other units (Fig. 4).

As a rule, for the reconstruction of blast furnaces one requires highly efficient machinery which should occupy as little space as possible. These requirements are met by 40-ton guy derrick or the "top" of the tower

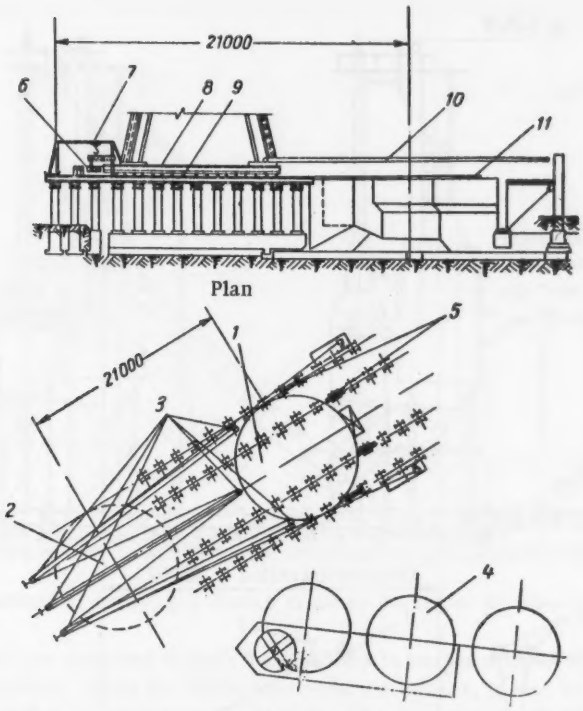

Fig. 3. Steel-constructed platform for shifting a blast furnace without lining:
1) place where the furnace is assembled; 2) final design position of the furnace; 3) rolling blocks; 4) blast-furnace stoves; 5) jacks; 6) hydraulic jack; 7) telpher for the transportation of supports and jacks; 8) rail assembly; 9) 144 mm diameter roller; 10) 30-line compound pulley; 11) slabs laid on the foundations.

crane of the same load-lifting capacity; they can be mounted on top of the blast-furnace stove (Fig. 5). By means of these cranes one can carry out the erection of the new furnace on the platform and the dismantling of the old furnace, the necessary materials being unloaded directly from or loaded onto flat railroad cars.

By a thorough preparation of the repair work, the removal of the "skull" in liquid state, carrying out as much work as possible before shutting down the furnace, a checked and large-scale erection and assembly of large steel units and sectional reinforced-concrete parts, a multistage and simultaneous carrying out of erection and assembly operations and a maximum mechanization of labor-consuming processes, it is possible to complete a 1st grade major overhaul of the blast furnace of 1386-1719 cu m in 28-35 days and a 2nd grade major overhaul in 12-13 days.

New constructional changes, introduced in the design of the standard blast furnaces, will have a marked effect on the reduction of the time required for the reconstruction and repair of furnaces as well on the improvement and the safety of working conditions.

At present, the designs of blast-furnaces include an air cooling system for the hearth bottom and a circular block of refractory concrete encircled by a steel ring which constitute the continuation of the furnace shell extended to the bed plate of the foundation. The bed plate is made flat without projecting parts; thus it is possible to transfer a furnace with its refractory brickwork and to reduce to a minimum the amount of work which has to be done after the shut-down of the furnace.

A particular feature of the standard designs of blast furnaces which can be rapidly erected, reconstructed

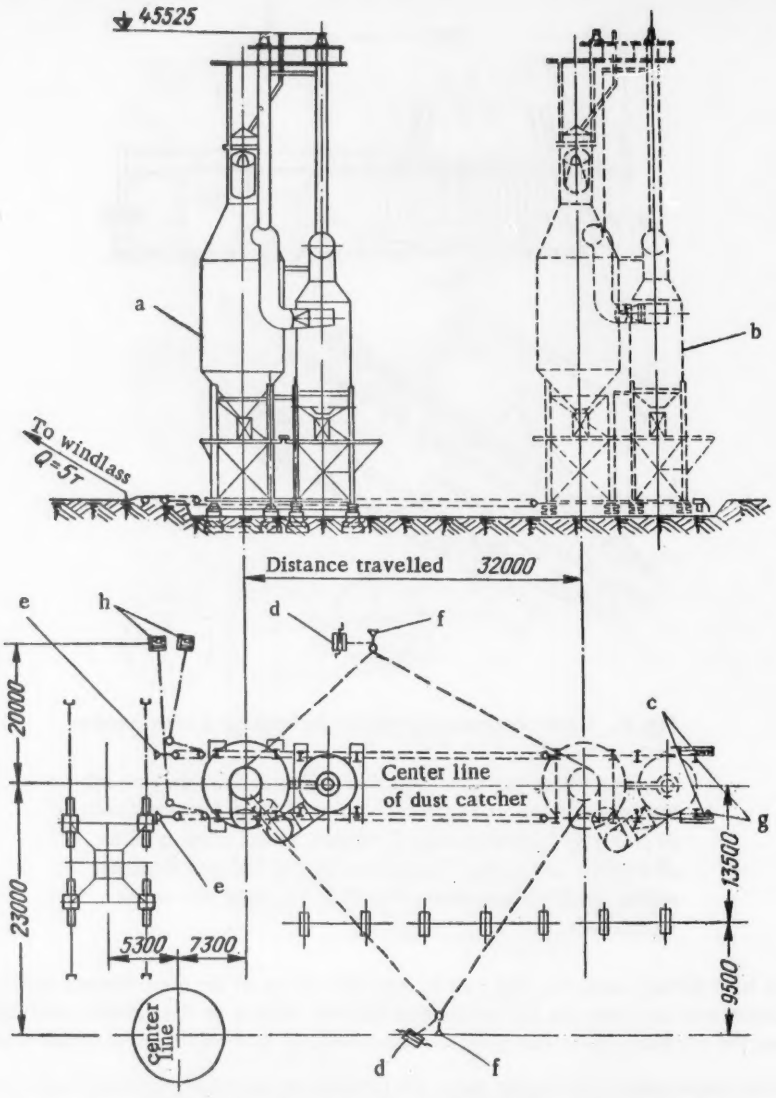

Fig. 4. Shifting of the dust-catcher unit: a) final design position of the dust-catcher unit; b) position where the unit is assembled; c) rail track for shifting the unit; d) 5-ton hand windlasses; e) 20-ton anchors; f) 5-ton anchors; g) 25-ton hydraulic jacks; h) PL-S-50 electric windlass.

and overhauled is an extensive use of sectional reinforced concrete which is used in the working floors of the blast furnace, the cast house, the blast-furnace stoves, the machine building and the enclosing walls of all the buildings just enumerated. Structures which consist of large assembled elements are so designed that the maximum possible number of parts are of standard dimensions.

The design of the supporting structures of the furnace makes it possible to rebuild or repair the bottom part of the furnace without the need for additional temporary attachments and equipment.

The installation of a central oxygen main from the oxygen plant or from a special station at the place of operation eliminates the delivery of oxygen in cylinders, and thus the working conditions and the safety of the work are improved.

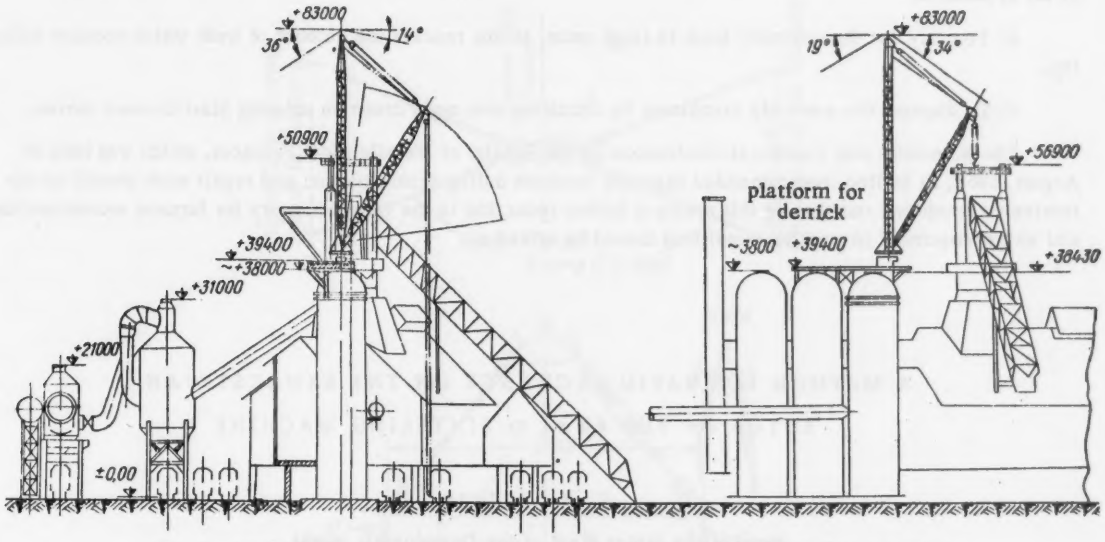

Fig. 5. Method of mounting a derrick crane on the top of the blast-furnace stove.

Refractory materials are delivered through the manholes in the gas offtakes and dust catcher as well as by the furnace lift; a monorail, under the bustle pipe round the furnace, is used for dismantling and assembling the tuyere equipment; debris is removed from the platform by means of boxes and tubes. All this speeds up and facilitates the repair work.

For rebuilding or repairing a furnace by rapid methods and for improving working conditions and safety, it is essential:

1. To prepare all materials structures and equipment beforehand, and to assemble them into large units;

2. To tap the "skull" when it is liquid, as in this way the labor-consuming and arduous operations on its removal when it has solidified are considerably reduced. Before the removal of the "skull" by blasting, it must be cooled.

3. To mark the sections and equipment of the furnace which have to be dismantled beforehand, as in this way accidental breakdowns and accidents are prevented;

4. To prepare the refractory materials thoroughly before the the shutdown of the furnace;

5. To cool the furnace well with a liberal supply of water before the dismantling operation; this promotes the breaking down of crust and facilitates the removal of the refractory brickwork of the shaft and hearth.

6. During a multistage operation, to use reliable protective covers, appropriately designed scaffoldings, ladders, protective covers, and other accessories which should be selected and approved before the commencement of the repair work; they should be inspected and acceptance records should be kept;

7. To set up a good communication and signalling system;

8. To disconnect the gas ducts and air ducts in good time (before the commencement of repairs) and to scavenge them as well as the dust catcher;

9. To feed fresh air into the furnace and, at the same time, to pump out contaminated gas from the furnace during the repairs;

10. To test the concentration of harmful impurities in the air at the places of operations systematically.

Since during the repairs and rebuildings various types of work take place at the same time and are carried out in several stages simultaneously, accident hazard becomes high and in order to prevent accidents it is necessary:

a) To eliminate manual work on labor-consuming and arduous operations by mechanizing these operations as far as possible;

b) To carry out the assembly work in large units, as this reduces the amount of work which requires climbing.

c) To disperse the assembly machinery by installing one main crane on existing blast-furnace stoves.

The Scientific and Technical Conference on the Repairs of Metallurgical Furnaces, which was held in August, 1958, in Stalino, recommended that new methods during reconstruction and repair work should be extensively introduced and that by this means a further reduction in the time necessary for furnace reconstruction and an improvement in working conditions should be aimed at.

A METHOD FOR RAPID EXCHANGE ON THE EXHAUST-FAN ROTOR OF THE 50 sq m SINTERING MACHINE

A. K. Rudkov

Head of the Sinter Plant of the Dzerzhinskii Works

The sintering machines at the sinter plant of the Dzerzhinskii Works are equipped with centrifugal fans of D-3500-13 type. The technical characteristics of the fan are:

Capacity, at intake conditions, cu m/min	3500
Rarefaction (vacuum), mm of water	1000
Temperature of the gas at inlet to the intake tube, °C	150
Specific weight of the gas at 0°C and 760 mm Hg, kg/cu m	1.33
Speed, rpm	1500
Electric motor capacity, kw	1300

The fan rotor is exchanged when the blades of the rotor are worn or when its vibration exceeds the permissible limit and cannot be eliminated.

Special tackle (Fig. 1) is used for a quick exchange of the rotor. The outfit for lifting the upper part of the housing (A) has two branches, each of which consists of a steel rope and a turnbuckle for adjusting the length of the branches and providing a uniform distribution of load. Both ends of the branch are joined into one ring which is suspended on the hook of the lifting mechanism.

The outfit for lifting the rotor (B) consists of a beam with two double hooks at the ends; two steel ropes which pass round the rotor and are attached to the double hooks of the brace; two lines for joining the brace to the hook of the lifting mechanism, one of the lines having a turnbuckle for adjusting the tension.

The equipment for lifting the shaft of the rotor (C) is used for raising the shaft slightly when the bearings are inspected.

It consists of a supporting frame, a double hook and a steel rope. The rope passes under the shaft and is attached to the double hook. When the hook is lifted, the shaft is lifted too and the bearing is exposed.

A stationary platform with railings, to facilitate and make safe the dismantling of the upper tube of the fan, is erected near the joint of the upper detachable tube of the fan and the tube from the multicyclone. Two lugs, which serve for lifting the upper tube of the fan, are welded to the tube at its center of gravity.

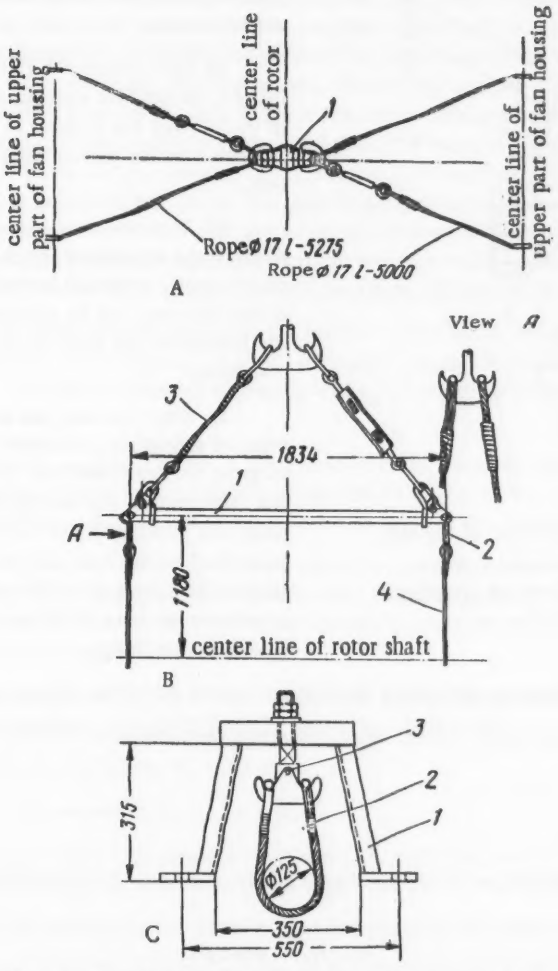

Fig. 1. Special equipment for rotor exchange: A) for lifting the upper part of fan housing: 1) double hook; B) for lifting the fan rotor: 1) beam; 2) double hook; 3) rope, 17 mm in diameter; 4) rope, 14 mm in diameter; C) for lifting the shaft of the rotor: 1) supporting frame; 2) rope, 13 mm; 3) hook.

Eleven men are employed on rotor exchange: 4 mechanics, 4 fitters, 1 electrowelder, 1 flamecutter and the operator of the crane. A foreman mechanic supervises the work. The operations are carried out in the following order:

a) After the fan has been stopped, the electric starting system of the fan is dismantled so that the fan cannot be started accidentally.

b) Bolts of the flange joint of the upper tube of the fan are removed and, at the same time, the upper housings of the bearings on the side facing the oil pump and the electric motor are dismantled and so is the cover of the clutch coupling. After the bolts on the flanges of the upper tube of the fan have been unscrewed, the tube is removed by means of an electric crane and lugs (Fig. 2). Then the covers of the bearing housings and the cover of the clutch coupling are taken off by means of the electric crane.

c) The bolts which join the upper part of the housing with the lower are unscrewed. The upper part is

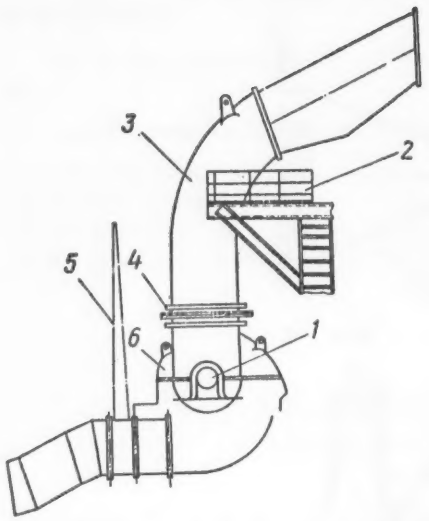

Fig. 2. Dismantling the upper tube of the fan: 1) fan; 2) platform; 3) upper tube; 4) expansion piece; 5) slide valve; 6) volute chamber.

lifted by means of the outfit A (Fig. 1) attached to the electric crane.

d) Two diffusers are dismantled, the rotor is removed, the contact surfaces of the upper and lower part of the housing are cleaned at the same time and a new asbestos cord for preventing air leaks and gas leaks is laid.

e) The new rotor is placed in position (by means of the same equipment which was used for the removal of the rotor). It should be pointed out that the bearings of the fans need not be adjusted for the new rotor since the journals of the shaft of all rotors are of the same diameter.

f) After the rotor has been mounted on the bearings, its alignment is checked by means of centering rings on the rotor shaft and the clutch coupling. For the alignment of the bearings there are five bearing pads with packing in the bearing race (three pads in the lower half of the race and two in the upper). In addition to the alignment of the packing, the bushings are fitted into the bore of the bearing housing with 0.04-0.08 mm stretching.

The remaining operations on assembling the fan are carried out in the reverse order.

The exchange of a rotor in the manner described above takes not more than 4-5 hours.

GAS-PURIFICATION OPERATIONS AT AN ELEVATED PRESSURE

M. N. Antsupov

Cherepovets Metallurgical Works

Blast furnaces of our Works operate at 1.5 atm pressure. The throttling valve unit at No. 1 blast furnace is located beyond the gas scrubbers and at No. 2 blast furnace behind the Cottrell precipitators. The housings of the scrubbers and of the precipitators were tested for tightness at 1.5 atm and for strength at 2.25 atm.

In the course of the pressure operation of the furnaces, there were no leakages at the flange joints, in the scrubber and Cottrell housings nor in the gas ducts. Graphitized gasket material, covered in copper foil, and ordinary gasket material saturated with red lead, was used for gaskets.

Some difficulties were encountered in the operation of the water seals of the Cottrell precipitators. Because the throttle valves at the water seals were leaky, the amount of water from the continuous washing of the precipitators was inadequate, and the water seals were blown when the gas pressure was 1 atm. It was necessary to cut off the water control seals and to bring in additional water to the seals; then the design of the throttle valve of the water seal was modified to provide a better tightness. After that, there was no need for extra water and there were no more blowings of the water seal by the gas.

At present, the gas pressure cannot be increased any further because of the limited height of the by-pass of the water seals of the scrubbers. During high-pressure operations, water flows over the upper baffle of the by-pass where there is no lock, and, if the pressure is accidentally raised to 1.5 atm, the water seal is blown out.

The valve floats fitted inside, as well as those fitted in separate outside chambers, are operating normally. The float chambers are washed periodically and, for this purpose, they have a water supply and an outlet to the slime chute. If the floats are inside the scrubber they are welded to the shaft so that they do not break away. In order to ensure a reliable operation of the throttle valve of the water seal at the high-pressure electric precipitators, the lever of the valve was made 250 mm longer. Electrified tapering gate valves of 1600 mm diameter, with blank flanges added, serve as the cut-off apparatus of the high-pressure electric precipitators. Since the valves are sufficiently tight, blank flanges are not fitted when the electric precipitator is shut down for repairs. CO content of the air inside the precipitator rarely exceeds 0.05 mg/liter. During the repairs, manholes, the flare and the slime outlet remain open. Before the work begins, air samples are taken for CO analysis. Although no difficulties were encountered when this procedure was followed, this method of shut down of the precipitator cannot be considered as normal; design organizations should develop a cut-off apparatus which would ensure a good seal without additional blank flanges, which are rather difficult to fit.

The construction of the water separator beyond the throttle valve unit of No. 1 blast furnace is unsatisfactory because the gas velocity is too high and, hence, all water separated in the throttle valve unit is carried away with the gas stream. In order to maintain the water seal of the water separator, it is necessary to add water into the lower cone of the seal.

At the gas outlet from the scrubbers of No. 1 blast furnace there are spray tubes of 600 mm mouth diameter. Fresh and reflux water is supplied for spraying at the rate of 50 cu m/hr. The pressure drop in the tube at a gas rate of 110,000 cu m/hr is 400 mm of water. The tube does not get clogged with slime.

If the blast furnace is operated with one scrubber, which has spraying tubes, the pressure head due to the sprayers causes difficulties in the operation of the furnace when it is changed to low pressure. It is necessary to design a by-pass with a throttle valve which could be opened to lower the pressure head of the gas every time the furnace operation is changed to low pressure.

When the pressure of the gas in the top of No. 2 furnace is lowered from 1.5 to 0.15 atm, more than 8000 cu m of extra gas enters the system, as is seen from the following calculation.

1. Gas pressure before the reduction, $P_1 = 2.5$ atm.

2. Gas pressure after the reduction, $P_2 = 1.15$ atm.

3. Volume of the vessel under high pressure (dust catchers, scrubbers, electric precipitators, gas ducts), i.e., the volume occupied by the given amount of gas at high pressure, $V_1 = 7000$ cu m.

4. The reduction of the pressure in the furnace top takes place at an almost constant temperature.

5. After the reduction of the pressure the volume of the given amount of gas will be

$$P_1V_1 = P_2V_2 \text{ and } V_2 = \frac{2.5 \times 7000}{1.15} = 15200 \text{ cu m.}$$

6. As a result of the reduction of pressure the volume of the gas increases by: 15200-7000 = 8,200 cu m.

At a total volume of the system equal to 8500 cu m (including the gas pipes of consumers) and at a total gas consumption equal to 3800 cu m per minute, the reduction of the gas pressure in the furnace top results in a marked increase in the pressure of the gas in the system (up to 3000 or more mm of water).

At a rapid change to high-pressure operation, the flow of gas from the furnace is drastically reduced for some time (owing to the compression of the gas) and, hence, a drop in the pressure of the gas in the system takes place.

There is a flare for burning the excess blast-furnace gas. Originally, the burners of the flare were equipped with control valves and nozzles which were later dismantled because of the low velocity of the excess-gas escape and their low efficiency. At present, the gas pressure in the system is controlled by a throttle valve with a servomotor. The output of the furnace increased from 90,000 to 130,000 cu m/hr at peak pressures and the speed of excess gas escape increased too.

However, even such an efficiency of the flare cannot damp the gas shocks at a rapid pressure drop of the

gas in the furnace top. Therefore, at excessive rise (or drop) in the pressure of pure blast-furnace gas a loud sound signal operates, and when the signal is sounded, the flares at the scrubbers or electric precipitators are opened by hand. Obviously, it is then necessary to have a mechanized, remotely-controlled (from the gas-cleaning control room) flare on the scrubber so that it would possible in the case of an accidental drop in gas pressure in the furnace top to throw off a large part of the gas under high pressure before opening the throttle-valve unit.

At the moment when the charge materials are poured from the large bell, the gas output from the furnace sometimes falls by 40%. To compensate for this reduction in gas flow rate, the boilers at the power station (the main consumer of the gas) are equipped with an automatic system which reduces the input of the gas by 40,000-50,000 cu m/hr in the course of a few seconds. At present, with the open-hearth furnace plant having been put into operation and with the increased volume of the gas system, the compensation for short-time shortage of gas has become easier.

No difficulties are encountered in cleaning the gas at a high-pressure operation of the furnaces. The dust carry-out from the blast furnaces which is accounted for constitutes 14 kg per ton of pig iron; final dust content is such that it is difficult to measure it by ordinary methods.

The dust carried out of the furnaces is fine and light and the purification of the wash water in the settling basins is inadequate. At an input rate of 0.47 cu m/sq m·hr, the wash water contains 480 mg/liter of suspended matter, and the purified water from the scrubber is 295 mg/liter. Some better methods of purifying the wash water in the settling basins must be found.

At present, in connection with the preparatory work on No. 2 blast furnace for a further increase in the gas pressure in the furnace top, the suitability of all units in the gas cleaning section are being checked. It was found necessary to carry out several modifications:

1) to increase the pressure of the reflux water;

2) to blank the bypasses of the water seals of the scrubber;

3) to increase the tightening loads on the flares of the scrubbers and electric precipitators;

4) to replace the throttle valves of the water seals of the scrubbers by tighter ones. This should provide normal conditions for gas-cleaning operations when the gas pressure is further increased.

NEW BOOKS

N. I. Krasavtsev

DEVELOPMENT PROSPECTS OF THE BLAST-FURNACE INDUSTRY

Moscow, Metallurgizdat, 1958, 558 pages.

N. I. Krasavtsev's book, which is to appear in the next few days, begins with a short historical survey of the development of the blast-furnace industry from the beginning of the XIXth century till the present time. This chapter contains a large amount of information which gives much detail of the development of the technology of pig iron production, the design of blast furnaces and their auxiliaries.

In the second chapter – "Ways for technical progress in the blast-furnace industry" – the author discusses in detail the problem of fuel for the blast furnace: a wider use of raw materials for coking, experiments on the production of "iron-coke," new coke-production methods by means of which metallurgical fuel can be obtained from badly coking or noncoking coals; he describes the effect of various factors on furnace output and fuel consumption; he tells about new methods of preparing materials for smelting and discusses a possible increase in the useful volume of the blast furnace. Separate sections in this chapter are devoted to technical innovations: the humidification of the blast, the pressure operation, the desulfurization of pig iron outside the blast furnace and the use of oxygen-enriched blast; and to the improvement of methods of blast-furnace process control.

In the last chapter of the book, the author has collected information on the methods of producing pig iron outside the blast furnace and on the prospects of future progress in pig iron production. The production of pig iron in electric, rotary and low-shaft furnaces is discussed and methods of obtaining the pig iron directly from iron ore are described. The chapter ends with the section called "Development prospects of pig iron production" in which the author, after a detailed survey of modern trends, draws the conclusion that it would be economical to replace the two-stage process of metal production by a multistage one.

The book is of great interest for technical personnel at iron and steel works and may be useful for university and technical school students.

A. G.

THE COMPLETE DEOXIDATION OF LOW-ALLOY STEEL IN THE LADLE

Engineers I. A. Sokolov, A. N. Vasil'ev and N. G. Nikulin

Kuznetsk Metallurgical Combine

09G2 and 09G2D low-alloy steels intended for light-section H-beam No. 19, 4-15 mm thick plate and other sections are made in basic open-hearth furnaces and cast by means of a stationary intermediate vessel.

For the purpose of establishing whether it is technically possible and economically practical to make steel without preliminary deoxidation in the furnace, 43 experimental heats were carried out.

In these heats, the addition of ferromanganese during the finishing period was omitted and the expensive metallic manganese was replaced by silicomanganese and ferromanganese. In other respects, the method of making and pouring the steel was left unchanged.

When the required content of carbon, phosphorus and sulfur had been achieved, the steel, while "boiling," was tapped into the steel ladle. No deoxidants were added to the furnace. The case samples for rapid analysis of steel and slag were taken at the commencement of the tapping. The introduction of deoxidants (in lumps of up to 50 mm diameter) into the ladle began when it was approximately one fifth full. First, silicomanganese and ferromanganese were added from a bin suspended on the pouring-ladle crane, then ferrotitanium was added from a stationary bin, and aluminum was charged by hand. In some of the heats a small amount of ferromanganese was added together with ferrotitanium.

Ferroalloys were added at a steady rate, without permitting separate lumps to stick together or to be covered with slag, and the operation was completed before the slag began to run from the furnace.

The amount of additions per 1 ton of steel was: 20-25 kg of silicomanganese, 2-2.5 kg of ferromanganese, 2.5-3.5 kg of metallic manganese (in six heats), 1.2-1.5 kg of ferrotitanium and 0.9-1 kg of aluminum; 45% ferrosilicon was not added to the ladle.

The time between the end of the tapping and the beginning of the pouring was 5-10 min. Steel was poured through an intermediate ladle with nozzles 33-40 mm in diameter, into 6.0-7.0 ton ingots.

For the study of the distribution of elements in the steel in the ladle, samples for chemical analysis were taken in 10 heats; after the fourth ingot was cast and then after every 4-6 ingots, 5-7 samples were taken for each heat, one sample always being taken after the last ingot had been cast.

The temperature of the metal in the furnace in the course of the heat was measured by a platinum-platinorhodium immersion thermocouple at the beginning, in the middle and at the end of the boiling period, 0-10 min before the deoxidation (ordinary heats) and 0-10 min before the tapping (for experimental heats). The temperature of the metal vefore the tapping in the experimental heats, and before the deoxidation in the ordinary heats, varied between 1630 and 1660° C.

The finished rolled products from the experimental heats were subjected to tests in accordance with technical specifications and GOST.

The length of separate periods of the heat, the consumption of the deoxidants and the quality of steel for experimental heats were compared with 14 ordinary heats (Table 1). The carbon content after the melt-down in the experimental and the ordinary heats was the same. The content of harmful contaminations (phosphorus and sulfur) during the melt-down was higher in the experimental heat than in the ordinary heats. When it was

TABLE 1

Comparison of Experimental and Ordinary Heats

Property	Heats	
	experimental	ordinary
Number of heats	43	14
Duration, hr-min:		
pure-boil period	1 – 25	1 – 19
deoxidation	–	0 – 21
finishing period, without deoxidation and tapping	2 – 40	2 – 41
tapping	0 – 08	0 – 08
finishing period, including deoxidation and tapping	2 – 48	3 – 10
finishing period	8 – 40	9 – 24
Content after melting-down, %:		
C	0.77	0.78
P	0.025	0.019
S	0.045	0.040
Content prior to oxidation, %:		
C	0.05	0.07
Mn	0.03	0.11
P	0.008	0.008
S	0.030	0.027
Mean (per heat) amount of Fe-Mn or metallic Mn added during the finishing period, kg	–	947
Rate of carbon removal, % C/hr:		
during the dephosphorization period	0.39	0.34
during the pure-boil period:	0.15	0.20
Temperature of steel before deoxidation (tapping), °C	1650	1645
Composition of finished steel, %:		
C	0.09	0.09
Mn	1.53	1.56
Si	0.35	0.28
P	0.019	0.024
S	0.027	0.023
Cu (09G2D steel)	0.29	0.31
Ti	0.02	0.02
Cr	0.04	0.04
Ni	0.05	0.05

to be fully deoxidized in the ladle, the steel was run from the furnace at a lower carbon content; this was done because of the higher carbon content in ferroalloys owing to the substition of silicomanganese (1.25-1.85% C) and ferromanganese (over 60% C) for low-carbon metallic manganese (0.10-0.13% C).

Out of 14 heats, deoxidized in the furnace and in the ladle, nine had ferromanganese added in the amount of 7.0-9.0 kg/ton during the refining period. As a result, the manganese content in those heats before the deoxidation was within the limits of 0.11-0.20%. Ferromanganese was not added to five of the ordinary heats and the content of manganese in the steel before the deoxidation was within 0.02-0.04%. All experimental heats

TABLE 2

Mechanical Properties of Steel from the Experimental and Ordinary Heats

Section	Heats	Number of samples	Yield point, σ_s kg/sq mm	Ultimate tensile strength, σ_b kg/sq mm	Elongation per unit length, δ %	Impact strength, a_k kg m/sq cm
With flange: up to 12 mm thick	According to TU	—	Not less than 35,0	47,0	18,0	—
more than 12 mm thick	—	—	Not less than 33,0	46,0	18,0	—
Beam No. 19 (light)	Exptl.	42	39,6	51,8	26,0	
	Ordinary	35	40,4	53,0	25,0	—
4-9 mm sheet	Exptl.	27	37,2	51,3	25,5	
	Ordinary	38	37,4	50,7	25,4	—
Angle 150×100×10 mm Channel No. 14	Exptl.	4	36,7	48,5	26,2	
	Ordinary	2	36,0	48,0	25,5	—
Shaped bars, and 11-24 mm thick sheet	According to GOST	—	30,0	Not less than 45,0	18,0	3,0
12-15 mm sheet	Experimental	293	36,6	50,0	23,3	6,7
Angle 130×130×14 mm	Ordinary	8	34,0	47,7	24,9	5,7

were carried out without the introduction of any manganese-containing additions during the finishing period and, hence the manganese content in steel prior to tapping was not higher than 0.05%.

A faster rate of carbon loss during the ore-boil period for the experimental heats than for the ordinary ones was observed. During the pure-boil period, the rate of carbon loss is higher for the ordinary heats, since steel from the experimental heats was tapped at a lower carbon content, when the rate of carbon loss is markedly reduced.

The temperature of the steel was 5° C higher for the experimental heats before the tapping than for the ordinary heats before the deoxidation.* Observations on the pouring of steel showed that, when the steel is to be fully deoxidized in the ladle, such an additional rise in the temperature of the steel ensures a quite satisfactory pouring operation.

Thus, there were essential differences in the technology of the experimental and the ordinary heats, the differences being caused by the change in the method of deoxidation and in the content of manganese in the course of the finishing period (conditions of the pouring operation were practically the same for the heats of both groups).

In spite of a higher phosphorus and sulfur content in steel after deoxidation and a slightly higher temperature of steel before the tapping, the duration of the finishing period, including the tapping, was shorter by 22 min for the experimental heats than for the ordinary ones (see Table 1). An increase in the duration of the pure-boil period for the experimental heats was compensated by a reduction in the duration of the period of

*According to the data from several steel works the temperature of the 09G2 steel when it is to be deoxidized in the ladle should be 15-20° C higher than that of ordinary heats.

TABLE 3

Consumption and Cost of Ferroalloys in the Experimental and Ordinary Heats

Ferroalloy	Composition of ferroalloy				+ Saving − Overexpenditure	
	in 14 ordinary heats		in 43 experimental heats			
	kg/ton of steel	ruble/ton of steel	kg/ton of steel	ruble/ton of steel	kg/ton of steel	ruble/ton of steel
Silicomanganese	16.70	19.00	22.90	26.00	−6.20	−7.00
Metallic manganese	10.10	49.00	0.46	2.23	+9.64	+46.77
Ferromanganese	4.30	4.30	2.47	2.47	+1.83	+1.83
45% Ferrosilicon	4.30	3.00	–	–	+4.30	+3.00
Aluminum	0.840	2.44	0.990	2.87	−0.150	−0.43
Ferrotitanium	1.45	5.41	1.40	5.22	+0.05	+0.19
Total	37.70	83.15	28.22	38.79	+9.48	+44.36

dephosphorization and slag making as a result of eliminating the addition of ferromanganese during the finishing period.

The steel from all experimental and ordinary heats satisfied the specifications of TU,* GOST and the requirements specified in the orders for the given steel grades.

It is seen from the data on the chemical composition of heats, as shown in Table 1, that when complete deoxidation is carried out in the ladle, the phosphorus content is reduced and the sulfur content remains approximately the same (taking into account its content after the melt down) as in the case when the deoxidation is carried out in the furnace and in the ladle.

The phosphorus content in the finished steel from the experimental heats is 0.006% lower than in the metal after the melt down. In the ordinary heats, on the contrary, it is 0.005% higher, because, when steel is fully deoxidized in the ladle, there is no liberation of phosphorus from the slag, which liberation takes place in the course of the deoxidation in the furnace. Although no ferromanganese was added during the finishing period and no preliminary deoxidation was carried out in the furnace, 0.001% more sulfur was removed from the experimental heat (during the finishing period, deoxidation and tapping) than from the ordinary heats.

The results of the chemical analysis of metal samples, taken in the course of pouring of 10 experimental heats, show that for all heats which were fully deoxidized in the ladle, a uniform distribution of carbon, manganese and silicon in the whole bulk of steel is achieved. Variations in carbon content for different samples amounted to only 0.01-0.02% i.e., were within permissible experimental errors. Only in some samples, taken at the beginning of the pouring, the content of manganese and silicon was found to be higher than the average for the heat. At the end of pouring, a fall in manganese and silicon content was observed and this can be explained by the interaction of the upper layer of steel with slag, resulting in the oxidation of manganese and silicon which pass to slag. As regards chemical composition, all samples, including those taken after the last ingots were cast (i.e., samples taken from the remainder of steel in the intermediate ladle), are acceptable, with an allowance for the deviations in manganese content. If, during the deoxidation of steel by the addition of ferroalloys into the ladle, the ferroalloys were not dissolved completely and in time and, hence, no uniform distribution of elements in the steel were achieved, the content of carbon, manganese and silicon in the last samples would be higher than the over-all value, but nothing of this kind observed in any of the experimental heats.

The quality of the ingot surface which is estimated by the method, acceptable at the Works, of visual inspection of ingots when they are charged into the soaking pits and when they are rolled on the cogging mill, was slightly better for the heats deoxidized fully in the ladle. The number of ingots with surface defects

*TU = Technical specifications.

detected during the charging of ingots into the soaking pits was 5% lower, and of those detected during the rolling was 2% lower for experimental heats than for ordinary ones.

Mechanical tests were made on specimens made from rolled shaped sections and rolled sheets as delivered. The data given in Table 2 show that mechanical properties of steel from experimental and ordinary heats are practically the same and are far higher than the specifications given by TU and GOST.

When all the deoxidants are introduced into the ladle at the time when the steel is tapped from the furnace, the losses of manganese and silicon are reduced because the burning losses of these elements are decreased and the possibility of ferroalloys getting mixed up in the slag is eliminated. Accordingly, the consumption of ferroalloys is reduced. The manganese loss in burning falls, on the average, from 30 to 20% and silicon loss from 46 to 20%.

The consumption and cost of alloys per ton of steel in the experimental and the ordinary heats are given in Table 3, from which it is seen that the total consumption of ferroalloys in the case of full deoxidation in the ladle is reduced by 9.5 kg/ton, and in particular there is saving of 9.6 kg/ton of metallic manganese, 4.3 kg/ton of 45% ferrosilicon and 1.8 kg/ton of ferromanganese. The consumption of silicomanganese increases by 6.2 kg/ton; the total cost of ferroalloy is reduced by 44 rubles 36 kop. per ton of steel. Owing to the shortening of the duration of the finishing period, the cost of making 1 ton of steel is reduced by 1 ruble 36 kop.

Thus, with the complete deoxidation of 09G2 and 09G2D steels in the ladle, the quality of the steel is not impaired and the production costs per ton of steel are reduced by 45 rubles 55 kop.*

At the Kuznetsk Metallurgical Combine, the complete deoxidation of 09G2 and 09G2D steels in the ladle without the introduction of metallic manganese and manganese additions into the furnace during the finishing period was adopted towards the end of 1957.

VARIATION OF THE HYDROGEN CONTENT IN THE COURSE OF A HEAT OF TRANSFORMER STEEL AND THE EFFECT OF GASES ON THE POROSITY (RISE) OF INGOTS

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It is known that when liquid steel solidifies its gas solubility falls rapidly. In particular, hydrogen solubility in transformer steel, which has the structure of silicon ferrite (Fig. 1), decreases markedly. Ferrite has body-centered cubic lattice, i.e., a more dense packing of atoms in the crystal than, for instance, austenite. In addition, the presence of silicon in steel has the effect of lowering the hydrogen solubility. As a consequence of the high gas content in liquid steel the rise of ingots is observed: a few minutes after the pouring, the steel in the mold begins to rise and overflows over the rims of the hot tops. Sometimes the rise of the ingots begins after a fairly long time has elapsed from the end of the mold filling.

Investigations which were carried out at a number of Works, showed that in the case of ingot rise, the outer zone in the upper part of the ingot is found to be affected by large blowholes (of honeycomb type) and the central zone has globular blowholes. When the steel crystallizes, hydrogen passes into the liquid steel in the ingot which becomes supersaturated and the hydrogen separates out, forming the gas phase. At as little as 7 cu cm/100g hydrogen content in steel, the evolution pressure of hydrogen approaches 1 atm.

*The main savings achieved are due to the elimination of the additions of Fe-Mn in the course of the heat and the replacement of the metallic manganese by silicomanganese, and this is not related to the method of steel deoxidation. Actual savings are made only on account of reducing the duration of heats (by 15-20 min) and lowering the losses of Si and Mn. Editor.

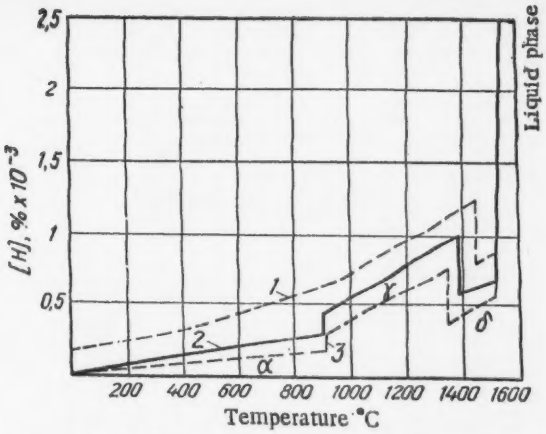

Fig. 1. Variation in the solubility of hydrogen during the solidification and cooling of austenitic and ferritic steels:
1) austenitic steel; 2) pure iron; 3) ferritic steel.

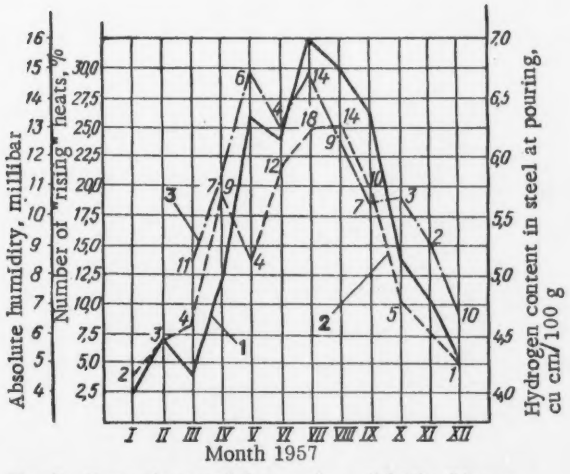
Absolute humidity, millibar
Number of "rising" heats, %
Hydrogen content in steel at pouring, cu cm/100 g
I II III IV V VI VII VIII IX X XI XII
Month 1957

Fig. 2. Dependence of the number of "rising" heats and the content of hydrogen in steel at pouring, on the absolute humidity of air (average monthly readings):
1) absolute humidity; 2) number of "rising" heats; 3) hydrogen content in steel at pouring (figures at the points indicate the number of heats).

The gas phase inside the liquid steel forces the steel up and breaks the weak solid-steel skin in the hot top. At the same time, the gases go into the atmosphere. In cases when the gas content is not too high, the ingot does not rise, but there is no shrinkage in the top part of the ingot.

For the purpose of establishing a quantitative relationship between the rise of ingots and the content of hydrogen and nitrogen in steel, samples were taken from the ladle for a large number of heats of transformer steel E3 after half of each heat had been poured. The method of smelting and pouring transformer steel at the "Dneprospetsstal'" Works has been described previously.*

In quenched specimens, kept in dry ice in Dewar flasks, the hydrogen was determined by evacuation on heating; nitrogen was determined chemically.

Table 1 shows the relation between the rise of ingots and the content of hydrogen and nitrogen in transformer steel. It follows from the data given in the table that all heats with up to 4.0 cu cm/100 g hydrogen content in steel exhibit ingot shrinkage in the mold. As a rule, the ingots cast from steel with a hydrogen content over 8 cu cm/100 g either rise or at least do not exhibit shrinkage. At intermediate hydrogen contents, the rise of ingots depends on the nitrogen content, the temperature of the steel and the rate of solidification.

The effect of nitrogen on ingot rise is much smaller than that of hydrogen, for in the course of the solidification process, only a very small amount of nitrogen passes from solution into gas bubbles which have already formed. The nitrogen content (3.6-9.6 cu cm/100 g) met in transformer steel, corresponds to a gas evolution pressure of 0.25 atm and hence nitrogen can pass only to an already formed gas bubble.

A significant relation was established between the total hydrogen and nitrogen content in steel (the total content varied from 7.5 to 19.5 cu cm/100 g) and the rise of ingots.

Apart from the effect of gas content, the investigation included the effect of temperature (as determined by a tungsten-molybdenum immersion thermocouple) before tapping on the rise of ingots. It is seen from Table 2 that with the increase in the temperature of steel before tapping, rising ingots occur in a large number of heats.

An increase in the temperature of steel promotes the absorption of gases from the atmosphere in the furnace because of the increased solubility of gases in steel. On the other hand, it retards the solidification of steel in the mold and increases the amount of evolving gases.

*G. M. Borodulin, G. F. Morenko and V. P. Frantsov. Metallurgist No. 3, 1956.

TABLE 1

Relation Between the Number of Rising and Nonshrinking Ingots, and the Content of Hydrogen and Nitrogen in Steel

Gas	Content, cu cm/100 g	Total number of heats studied	In this number:					
			heats with raising ingots		heats with nonshrinking ingots		total heats with rising or nonshrinking ingots	
			number	%	number	%	number	%
Hydrogen	Up to 4,0	3	—	—	—	—	—	—
	4,01—5,0	18	2	11,1	1	5,5	3	16,6
	5,01—6,0	21	4	19,0	3	14,3	7	33,3
	6,01—7,0	18	3	16,7	4	22,2	7	38,9
	7,01—8,0	9	3	33,3	1	11,1	4	44,4
	Over 8,0	7	3	42,8	3	42,8	6	85,6
Nitrogen	Up to 4,0	11	—	—	2	18,1	2	18,1
	4,01—4,80	14	3	21,4	1	7,1	4	28,5
	4,81—5,60	10	3	30,0	2	20,0	5	50,0
	5,61—6,40	3	1	33,3	1	33,3	2	66,6
	Over 6,40	15	4	26,7	4	26,7	8	53,4
Nitrogen + hydrogen	Up to 11,0	25	1	4,0	1	4,0	2	8,0
	Over 11,0	15	4	26,7	4	26,7	8	53,4

TABLE 2

Relation Between the Number of Rising and Nonshrinking Ingots, and the Temperature of Steel Before Tapping (all heats were evacuated in the ladle)

Temperature of steel before tapping, °C	Total number of heats studied	In this number					
		heats with rising ingots		heats with nonshrinking ingots		all heats with either rising or nonshrinking ingots	
		number	%	number	%	number	%
Up to 1600	19	1	5.2	3	15.8	4	21.0
1601-1620	48	5	10.4	9	18.8	14	29.2
1621-1640	72	11	15.3	20	27.7	31	43.0
1641-1660	35	8	22.8	6	17.2	14	40.0
Over 1660	12	7	58.2	2	16.7	9	75.0

One of the main sources of hydrogen in steel is the moisture in air. It is known that the solubility of hydrogen in liquid steel is proportional to the partial pressure of hydrogen in the atmosphere above the steel:

$[H] = K\sqrt{p_{H_2O}}$.

Figure 2 shows the relation between the number of heats with rising ingots, mean hydrogen content in steel for a given month and the average monthly absolute humidity of air (from the reports of the Zaporozhe Meterological Station). An increase in the hydrogen content in steel during the summer months is found after

the melt-down of the charge, when the contact of steel with the atmosphere in the furnace is easiest. In the summer, a relatively high hydrogen content in steel is retained until the tapping and, therefore, in order to reduce hydrogen content at that period, it is necessary to have a dry atmosphere in the furnace.

In the course of the study, the behavior of hydrogen in steel during the oxidation period was investigated. The hydrogen content in steel of 20 heats decreased during the boil period from 5.00 to 4.86 cu cm/100 g on the average, and, at the same time, in seven heats an increase in hydrogen content at the end of the boil period was detected. Poor degasification of steel during the oxidation period was caused by the blowing of the steel at the end of the boil with inadequately dry oxygen.

In steel from 14 heats, the hydrogen content was determined before and after the oxygen blow. It was found that the hydrogen content in steel increases after the blow by 0.26 cu cm/100 g on the average. Determinations of oxygen humidity by means of a hygrometer showed that the moisture content per 1 st cu m of oxygen constituted approximately 3-6 g.

As a rule, alloying the steel with silicon caused a marked increase in hydrogen content in steel. On the average, for 51 heats, the hydrogen content increased in 15-20 min after the addition of 75%-ferrosilicon by 0.9 cu cm/100 g (to 6.4 cu cm/100 g).

For several heats, the hydrogen content in 75% ferrosilicon was determined by means of evacuation by heating. The hydrogen content in the batches of ferrosilicon varied from 5.0 to 16.3 cu cm/100 g. When ferrosilicon of a high hydrogen content was added to steel, the hydrogen content of steel was up to 3 cu cm/100 g.

An effective method for the degasification of steel after discharge from the furnace is the vacuum treatment of steel during the transfer from one ladle to another. It is possible by this method to remove on the average 12 cu cm of hydrogen per 100 g of steel and to reduce markedly the number of heats with rising ingots. Out of 117 heats of transformer steel treated under vacuum (the final pressure in the evacuation chamber was 25-60 mm of Hg) in 1958, only two heats (less than 2%) had rising ingots. This method of steel treatment is adopted in the production of transformer steel at the "Dneprospetsstal'" Works.

Hence, in order to obtain nonporous ingots of transformer steel, it is necessary, apart from roasting the materials used in the heat, to use dried oxygen and low-hydrogen-content ferrosilicon and to transfer the steel from one ladle to another under vacuum. The establishment of dry atmosphere in the furnace by ensuring that only dry compressed air enters the furnace, will make it possible to obtain steel with an even lower hydrogen content. Toward this end, a large research program is under way at the Works.

ON THE STRUCTURE OF RIMMED-STEEL INGOTS

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"Azovstal" Works

Rimmed steel of up to 0.37% carbon content constitutes approximately 60% of the output at the "Azovstal" Works.

Until recently, it was thought that it was impossible, or at least uneconomical, to make ordinary-quality, silicon-free steel of 0.28-0.37% carbon content.

Experience has proved this opinion to be unfounded. Such a steel (with 0.7-1.0% manganese content) has been made and used for rolling mine props at our Works since 1955. Products made of this steel fully satisfy GOST specifications.

The steel is made from phosphorus-containing pig iron in tilting open-hearth furnaces which are fired with mixed gas combined with the use of oxygen. Rimmed steel is bottom-poured into open, big-end-down molds.

Operating Statistics of Making and Pouring Rimmed Steel at the "Arovstal" Works*

Item	Heat number			
	2274	11285	5248	10279
Steel grade	Ometiz	Ometiz	3kp	3kp
Duration, hr-min:				
heat	12-45	13-35	15-00	13-05
ore-boil period	1-40	2-25	2-10	2-50
pure-boil period	1-00	1-00	0-45	0-45
deoxidation	0-15	0-15	0-20	0-15
tapping	0-15	0-15	0-15	0-15
Composition of metal after the melt-down, %:				
C	0.73	0.60	0.70	0.88
Mn	0.05	0.19	0.08	0.22
S	0.063	0.068	0.063	0.07
P	0.025	0.13	0.048	0.235
Rate of carbon removal in burning, % C/hr:				
during ore-boil period	0.34	0.18	0.21	0.18
during pure-boil period	0.06	0.07	0.13	0.12
Composition of steel (ladle sample), %:				
C	0.10	0.07	0.22	0.36
Mn	0.30	0.47	0.42	0.71
S	0.052	0.038	0.049	0.03
P	0.036	0.03	0.032	0.038
As	0.135	0.112	0.140	0.138
Time of mold filling, min-sec	6-05	6-28	5-50	5-25
Time of rimming action in the mold, min	30	15	15	3

*Steel from all heats was cast into 6.8 t ingot; molds were filled to a height of 2160 mm.

The replacement of killed steel MSt.5 by rimmed steel (or, more accurately, semikilled) makes the preparation of the molds cheaper and quicker, and results in a saving of about 10% in steel. Steel consumption for 1 ton of useful rolled product from 5kp steel is the lowest, as compared with other rimmed steels. This fact is confirmed by the steel consumption coefficient recorded at the blooming mill (in subsequent stages of steel processing no separate records are kept for each heat and steel grade, but the loss of steel in the course of conditioning and inspection is small and constitutes about 3-5 kg/ton).

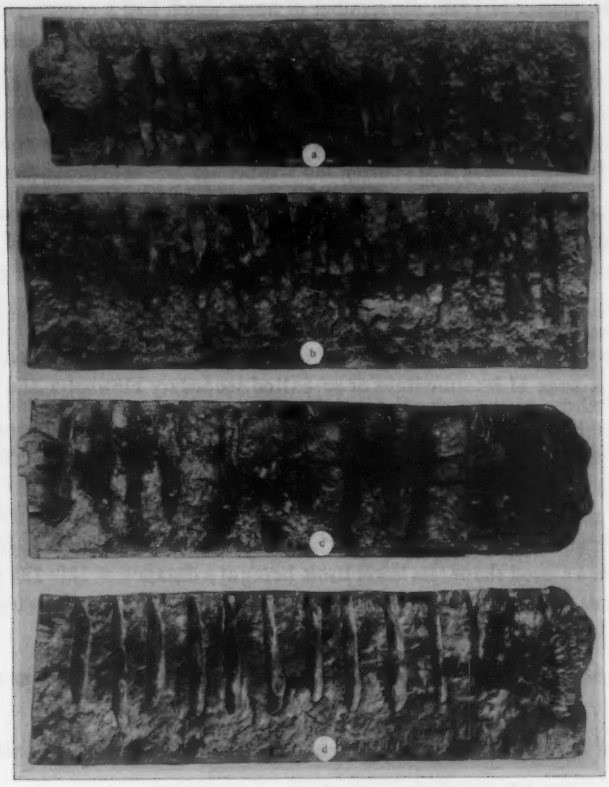

Fig. 1. Longitudinal axial fracture cross section of ingots from rimmed steel of various grades:
a,b) Ometiz; c) St.3kp; d) St.5kp.

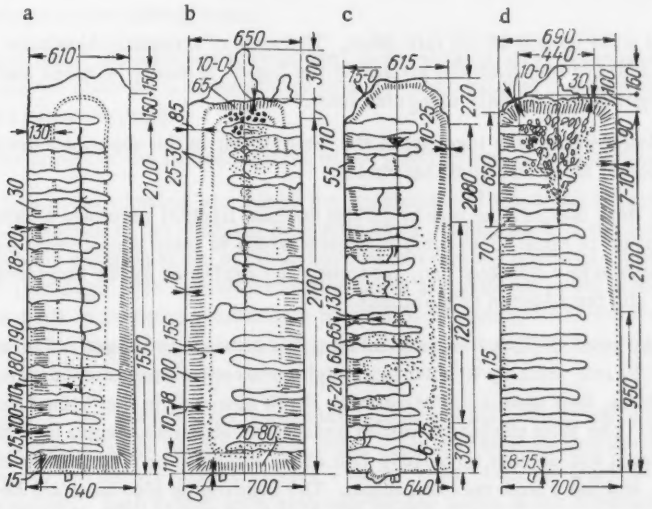

Fig. 2. Sketches of longitudinal axial fracture cross section of ingots from rimmed steel of various grades:
a,b) Ometiz; c) St.3kp; d) St.5kp.

Thus, in the second quarter of 1958, the number of heats which had a consumption coefficient less than 1.119 constituted 53% for steel grades Ometiz, MSt. 1, MSt.2; 65% for steel grade MSt.3 and 86% for MSt.5kp.

It should be pointed out that since February 1958, in connection with the change to the deoxidation of rimmed steel with ferromanganese (containing 2.0-3.0% Si) in the ladle, the consumption of 5kp steel at the blooming mill fell, more than 80% of the heats of 5kp steel having a consumption coefficient below 1.119. The table gives operating statics on making and pouring the steel.

It should be mentioned that the ingots of 5 kp steel which involve less metal loss when prepared for rolling, are, in contradistinction to ingots of other steels, obtained when they are allowed to solidify freely i.e., without the application of the cap at the top of the mold and without the addition of aluminum to the top end.

It is interesting to compare the structure of ingots from various grades of rimmed steel (Figs. 1 and 2).

The ingot of Ometiz steel (Figs. 1a, and 2a) exhibits a solid skin zone, a honeycomb blowhole zone, an intermediate solid zone, a zone of intermediate or so-called secondary blowholes and a core.

The solid core zone is 15 mm thick near the gate and 10-15 mm at the wall near the bottom. Above 1550 mm up the ingots, the thickness of this zone increases from 8-20 to 130 mm. The honeycomb blowhole zone extends through 1550 mm of the ingot height, i.e., it occupies 2/3 of the height of the ingot. The size of honeycomb blowholes gradually decreases: the length of honeycomb blowholes is 100-110 mm at the bottom end of the ingot and 30 mm at a height of 1500 mm. The intermediate solid zone is 50-80 mm in width. The zone of secondary blowholes has a closed shape, its lower limit being at a distance of 180-190 from the bottom surface of the ingot and the upper limit 130 mm from the top surface of the ingot. The core of the ingot has, in the main, a solid structure, except for the lower part where small blowholes are found.

As far as the internal structure is concerned, the given ingot can be regarded as a satisfactory ingot of rimmed steel.

The second ingot of the same steel grade from another heat (Fig. 1b, Fig. 2b) also exhibits five zones.

The solid skin zone of 10-18 mm thickness in the lower part of ingot increases gradually and is up to 95 mm thick in the upper half of the ingot. There is no solid skin in the bottom and the top ends of the ingot.

The zone of honeycomb blowholes extends over the whole peripheral part of the ingot and is of a closed shape. In the lower half of the ingot, the length of the blowholes is 70-80 mm at the bottom end and 100 mm at the walls of the ingot; in the upper half of the ingot, the length is 25-30° at the top end and 65 mm at the walls.

The intermediate solid zone is 35-50 mm thick. The zone of secondary blowholes has a closed shape and is at a distance of 155-170 mm from the surface of the ingot. It should be noted that secondary blowholes in the top end of the ingot have a slightly elongated form.

The core in the lower half of the ingot is, in the main, solid; in the upper half there is porosity in the form of a conglomeration of large and small blowholes.

This ingot differs from the previous one by its less uniform internal structure: a deeply spread shrinkage porosity and a large amount of large honeycomb blowholes which in some places come out to the surface of the ingot. Such an unsatisfactory structure can, apparently, be explained by the increased manganese content (0.47 instead of 0.30%) and the addition of aluminum.

The ingot from 3kp steel exhibits the same zones as the previous two ingots. The solid skin zone is 6-25 mm thick near the gate, 15-20 mm thick up to 1500 mm height; above the level of 1500 mm, the thickness of the sound skin rapidly increases, then gradually decreases to 10-25 mm, and in the "risen" part of the ingot the skin is no longer quite sound. The zone of clearly formed honeycomb blowholes begins at a distance of 300 mm from the bottom end of the ingot and extends up to 1200 mm above the bottom; then it starts again, but from the zone of secondary blowholes, and has a half-enclosed shape. The honeycomb blowholes are 55-65 mm long.

Fine, not fully developed honeycomb blowholes are found in the bottom part of the ingot up to 300 mm from the bottom. The intermediate zone occupies only 40-45 mm. The secondary blowhole zone extends only over 2/3 of the ingot height and is at a distance of 130 mm from the ingot surface.

In its upper third, the ingot core has a well pronounced shrinkage porosity which passes into a more or less defined conglomeration of gas blowholes in the remaining part of ingot.

With regard to its internal structure, this ingot would be regarded as a not quite satisfactory rimmed-steel ingot. The deterioration in the ingot structure is apparently connected with the addition of aluminum to the ingot top.

Unlike the preceding ones, the ingot of 5kp steel (Figs. 1d, and 2d) has only three zones: the sound skin zone, the zone of honeycomb blowholes and the core.

The thickness of the sound skin zone is as follows: 15 mm in 2/3 of the ingot height, 7-10 mm in the upper third and 30-0 mm in the top end. The zone of clearly defined honeycomb blowholes begins at a distance of 950 mm from the bottom of the ingot. The honeycomb blowholes are up to 70-90 mm in length. Short, undeveloped honeycomb blowholes are found in the lower part of the ingot (up to a height of 950 mm). The core of the ingot is in the main sound except its upper third which has large enclosed shrinkage porosity consisting of shrinkage holes and blowholes.

With regard to its internal structure, this ingot belongs to normal ingots of semikilled steel.

The study of the structure of rimmed-steel ingots shows that if the main points of the technological instructions on making and pouring the steel are observed, it is possible to produce sound ingots which will ensure that the consumption of steel per ton of useful steel for rolling is at a minimum. The quality of ingots of 5 kp steel is such that is is possible to extend the application of this steel in the national economy.

IMPROVED TRACK SYSTEM IN OPERATION AREAS OF OPEN-HEARTH PLANTS

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With a substantial increase in productivity, most of the open-hearth plants experience frequent delays, in particular during furnace charging, due to delays in the delivery of the charge materials. These delays are caused mainly by transport difficulties due to the continuously increasing volume of materials carried by the existing system of trackage in the working areas.

The main shortcoming of the trackage systems used in the working areas is the presence of a large number of crossings which do not allow for a rational development of the existing trackage and which reduce the carrying capacity of the railroad system.

Figure 1 shows the layout of the railroad system in the working area of a modern open-hearth plant.

As is seen from Fig. 1, the crossings are due to the intersection of No. 3 and 4 tracks by long crossovers which link No. 1 unloading track of the charging floor with No. 5 middle track of the lean-to. Consequently, in such a system, the loaded charging boxes going to the furnaces and the returning empty boxes have to be carried via No. 5 track (running). The cars with charging boxes are gathered in the lean-to on No. 6 track only. The use of No. 4 track of shunting operations is made difficult because of several crossings which, in fact, not only increase the number of angle runs during shunting operations with charging boxes, but also reduce the carrying capacity of the lean-to tracks.

The railroad system of the working area open-hearth plant and the lean-to are operated under very difficult conditions (constant traffic, large loads on axles, etc.). This is even more aggravated by the fact that the working-area tracks, which are not gravelled and have a fairly "rigid" bed, are provided only with switch units with straight-line components. Such switch units have two turning points in the direction from the straight switch to the crossover curve and from the curve to the straight crossing (frog). At these turning points, i.e., at the point of the switch heel (Fig. 2a) and the point of the frog, there occur, in addition to jerks which upset smooth movement, the most intense disruption of the track and a pronounced wear of the parts of the switch and of the running parts of the rolling stock. All this results in frequent derailments and, hence, delays in the delivery of charge materials.

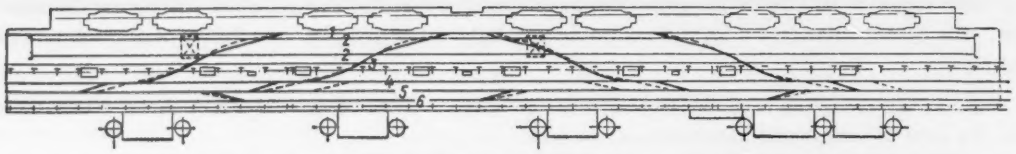
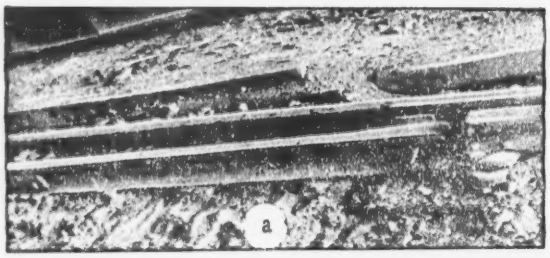


Fig. 1. Layout of rail tracks of an open-hearth shop: 1) unloading (charging track); 2) charging-machine track; 3) track for hot-metal ladle cars; 4) 5) 6) lean-to tracks (dotted lines indicate crossovers which are laid in a modern open-hearth plant).



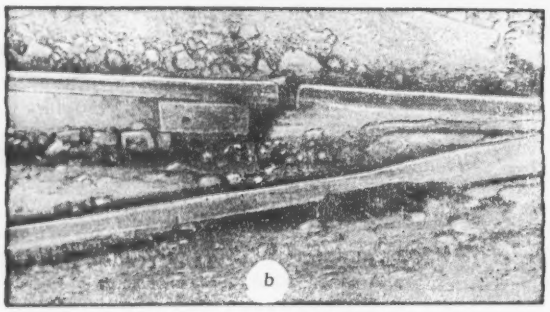


Fig. 2. Switch heel (a) and frog (b) of the switch (Zakavkaz Metallurgical Works).

Because the track sections on the charging side are short and the speed of the small-wheel-base rolling stock is slow, the shortest and most compact junction units of the railroad lines have to be installed. The compactness of the parts is achieved, in the main, by the application of curves crossings, curvatures of small radii, short switch blades and sharp frogs.

In this respect the application of special units as: symmetric switches, double switches with single and two-side curvature, single switches, intersection switches with curved parts, etc., is very effective.

Under the existing limited-space conditions in the working area the railroad system can be improved and the disadvantages mentioned above can be eliminated by introducing curved switches, *i.e., switches with curved switch blades and frogs.* Such switches, unlike those with straight-line parts, cannot be universally used for both right-hand and left-hand turnouts. However, many years experience of the application of such switches for town tracks as well as in industrial and main railroad lines in foreign countries (e.g., at iron and steel *works in Germany) shows that these switches have* several advantages as compared with ordinary switches.

Curved switches which have no rectilinear parts ensure a softened entry angle of the impact and improve *the movement of the rolling stock over the branching track, since the turning points on the passage from the* straight switch to the crossover curve and from the crossover curve to the straight frog are eliminated. Hence, sharper-curve crossovers of shorter length and turning a larger angle can be applied, and, at the same time, smoother entry conditions of the rolling stock into the switch are achieved. By using such switches, it becomes possible to produce more compact schemes of track layout, and, in addition, the curved switches reduce the local wear of switch components and of the running part of the rolling stock.

It is advantageous to use composite curved switches in the working area. Similarly to the composite switches which are laid at the open-hearth plants and consist of ordinary type 1/7 switch blades and type 1/5 frogs, the curved switch consists of 1/16 curved switch blades (equivalent to 1/7) and 1/4 curved frogs (equivalent to 1/5).

As the basis for the equivalence of such a curved crossover the following conditions were assumed in their design: a) the maximum and the minumum impact angles into the curved-switch blade at the entry to the turnout should be smaller than the impact angle into a straight-switch blade; b) the curvature radius of the curved blade should be not smaller than that of the of the crossover curve; c) the curvature radius of the crossover curve and of the frog in the "equivalent" crossover satisfy the requirements specified for the entry of the rolling stock

Shunt Characteristics

Characteristic	Composite switches	
	with straight parts	with curved parts
Length of the switch blade, mm	4500	4000
Maximum angle of impact on the blade	1°48'30"	1°42'47"
Distance from the beginning of the stock rail to the center of the switches, mm	9231	8234
Distance from the center of the switch to the end of the frog, mm	8819	7024
Total length of the switch, mm	18050	15258

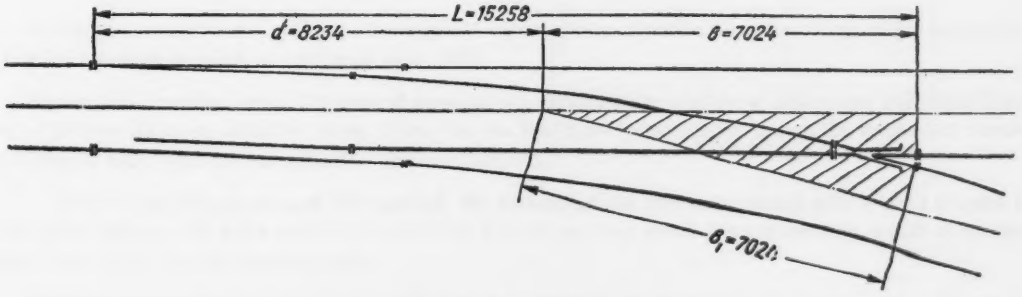


Fig. 3. Line diagram of the composite curved switch.

operating on the charging side 9p series locomotive and charging box cars of 50 t capacity, 4.84 m long, and 2.3 m wheel base.

The line diagram of the composite curved switch is shown in Fig. 3. The advantages of this switch over the corresponding composite switches which have rectilinear elements are indicated in the table.

As is seen from Fig. 1, the application of the curved switch (continuous lines) provides for the crossover to ajoin No. 4 line of the lean-to, instead of No.5 as at present (dotted line). In this way, it is possible to eliminate the crossing in the lean-to and to include No. 4 line in the operations.

By the use of the curved switches in the whole layout of the track system on the charging floor, all four crossings in the lean-to are eliminated and, hence, the time of shunting operations with loaded and empty charging box cars can be reduced. The time is reduced on account of the simultaneous arrival (or dispatch) of the cars from the stock yard and the delivery (or removal) of the cars for unloading in the furnace bay.

The reduction in the time spent on shunting operations is further improved by the reduction in the length of shunting runs in transferring cars from No. 4 line to the furnace bay, since there is no need for a preliminary shunting of the cars from No. 4 line to No. 5 line. A punctual delivery of cars to the furnaces is ensured by bringing loaded cars onto appropriate sections of No. 4 line beforehand independent of the extent to which Nos. 5 and 6 lines of the lean-to are actually occupied.

This method of fully employing No. 4 line in the lean-to fits very well into the improved system of ferroalloy delivery to the furnace bay and the mechanization of furnace repairs. The removal of ferroalloy storage from the lean-to, the application of belt conveyers for the removal of debris, and of self-loading cars for the delivery of refractories make No. 4 line actually free of cars and make it possible to use this line mainly for charging box cars.

In addition, the linking of No. 1 line of the furnace bay with No. 4 line increases the useful capacity of the lean-to. Two sets of charging box cars, instead of one, can be accomodated on the middle section of this line, since two crossovers are usually laid between No. 4 and No. 5 lines (on Fig. 1 these crossovers are shown by a dotted line). The small overall length of the curved switch and the reduction in the distance to the end point of a given switch also increase the useful capacity of each section and of the lean-to.

The application of curved switches improves the entry of the rolling stock owing to the improved entry impact angle (the maximum angle of impact on to the switch blade is 1° 42' 47", the minimum 1° 15' 00") and the elimination of turning points on the turnout.

In conclusion, it should be pointed out that the application of the curved switches is not limited to the improvement of the existing track systems on the charging side. The curved switches can also be used for an economic track layout in other sections of the open-hearth plant (stock yard, pit side, etc.).

The problem considered here is of special significance also in the development of new designs for open-hearth plants with block layout of open-hearth furnaces, because the application of curved switches in those plants will not only help in the unification of all switches, but will also provide a more compact railroad scheme and a better general layout of the whole site of the open-hearth plant.

ROLLED AND TUBULAR PRODUCTS

INCREASING THE EFFICIENCY OF THE BLOOMING MILL

M. I. Stukalov

Head of the Blooming Mill at the "Azovstal" Works

The one-stand two-high reversible blooming (primary) mill, which serves the rail-structural and the heavy-section mills, rolls billets of the following dimensions: 270 × 280; 280 × 300; 330 × 340; 245 × 280; 320 × 140 × 375; 320 × 120 × 400; 230 × 280; 208 × 220 mm.

Owing to improvements in the technology and organization of production, the output of the blooming mill in 1958 almost doubled as compared with 1950.

The mill is provided with 10 groups of soaking pits, fired with a mixture of coke-oven and blast-furnace gases of 1600 kcal/cu m calorific value. Here, for the first time in the Soviet Union, the method of removing slag in liquid state from the recuperative pits was adopted.

Prior to the introduction of this method, the bottoms of the pits were fettled with a layer of coke fines which, upon mixing with scale and other impurities, formed dry slag which was removed by means of special shovels after every third or fourth charge.

With the introduction of the method of liquid slag removal, the bottoms were fettled with a 50 mm deep layer of boiler slag after every 10 or 15 charges. For a better slag formation and in order to avoid the clogging of the bottom, particles over 20 mm in size were separated from the boiler slag in a rotary drum.

In the course of furnace operation the boiler slag, mixed with scale, forms a mobile fluid mass which flows from the heating chamber through a cinder notch into a ladle. Data on the composition, softening temperature and melting temperature of the slag formed are given in the table.

The adoption of the continuous removal of slag made it possible to reduce the number of furnace fettlings by a factor of 3 or 4, to raise the operating temperature in the heating chambers by 30°C, to reduce the heating time of ingots and to increase output by 20%.

The saving on the coke fines (16,000 tons) alone exceeded 800,000 rubles per year.

At the same time, an extensive program with the object of increasing the weight of ingots was carried out at the Works. The weight of rimmed-steel ingot was increased by 0.8 tons to 6.8-7.0 tons; the weight of rail ingots was increased by 6.6 to 9.75 tons.

At present, 8-ton rimmed-steel ingots are being introduced and are rolled without any increase in the number of passes.

With an active participation of the innovators of the mill, the cathodic protection from corrosion of roll necks when cooled with sea water, was introduced. The life of texolite bearings increased from 5-6 days to 4 months, the number of roll changes (due to the breakdown of bearings) decreased from 6 per month to 3 in the course of two months.

In addition, the drive unit of the mill rolls was modified. Now, the rolls are driven directly by four motors without intermediate reducing gears.

The speed of the ingot buggy was increased from 3.9 to 50 m/sec, by means of changing the slopes and

Composition (%), Softening Temperature and Melting Temperature of the Boiler Slag Introduced and of the Slag Discharged from the Pit

Slag	Fe	FeO	F_2O_3	MnO	Al_2O_3	SiO_2	CaO	MgO	P_2O_5	C	Temperature, °C	
											softening	melting
Before the introduction of boiler slag	6,0	5,12	2,85	0,45	5,25	11,30	2,50	0;90	0,31	63,50	1310	1350
Flowing from the pit	53,2	54,82	15,28	1,3	2,74	24,2	2,0	0,40	0,09	—	1380	1250—1280

replacing the 75 hp motor by a 150 hp one. However, the speed of the buggy again became inadequate in relation to the attained output of the mill.

When the 9.75 ton ingot was introduced, an attachment for counterbalancing the buggy-tilting mechanism was designed and constructed, and the durability of the screwdown mechanism was improved by increasing the height of the thread and by cooling the screw housings of the screwdown mechanism.

The time for reversing the main drive was reduced from 7 to 5.5 sec.

Owing to the introduction of these technical improvements, the amount of defective production which was caused by welders and mill operators decreased markedly. Thus, in 1950, defective production constituted 0.22% of the accepted rolled products; in 1955 it constituted 0.063%, and in 1957 0.049%.

At present, the personnel of the blooming mill is working on improvements in the durability of refractories of the soaking pits, the introduction of 8-ton ingots of rimmed steels, a further increase in output per hour and an improvement of the machine for hot-marking of ingots while they are moving.

EXPERIENCE ON THE PRODUCTION OF SECTIONS FOR MINING CONSTRUCTIONS

N. F. Protasov, V. P. Khlebnikov, A. I. Sikorskii, V. V. Gonchar, L. I. Boldyrev and V. E. Stefanov

"Azovstal" Works

The "Azovstal" Works is one of the biggest producers of main sections used in the construction and operation of mines. Some of these sections – mine props of two types, 18A – 18B and 28A – 28B, are shown in Fig. 1.

For these sections to function, it is necessary for section B to enter partly into section A, thus forming a permanent light joint (Fig. 2). Hence, the outer side walls of the B props and the inner walls of the A props constitute operational surfaces and should not have any traces from roll work.

The 18A – 18B and 28A–28B mine props are rolled from 230 × 285 and 245 × 280 blooms in a four-stand arranged in two lines. In the roughing stand, the piece is passed four times in the first pass, twice in the edging pass and once in the channel-forming pass.

The rolls of the roughing and the first stand are made of grade 55kh forged steel. For increasing the life of the rolls, the parts which are most exposed to wear are hardened. Rolls of the second and the third stand are of cast iron of 280-320 Brinnell hardness units.

The rolling of mine construction sections was carried out according to a definite method developed at the "Azovstal" Works.

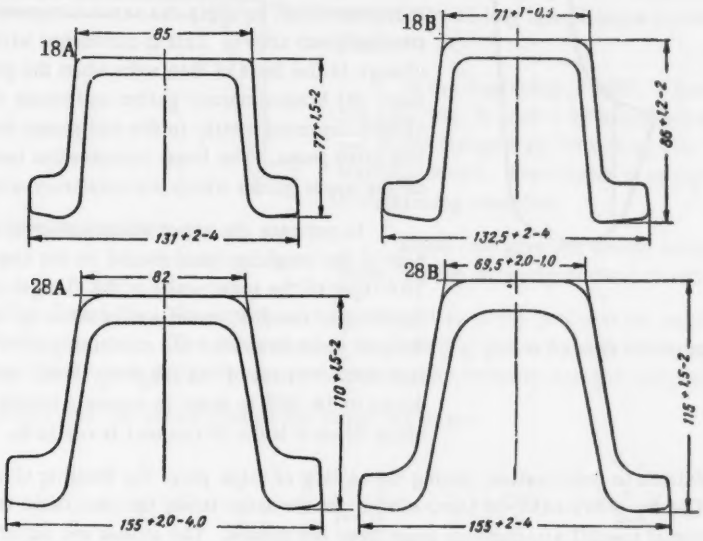


Fig. 1. Props 18A–18B and 28A–28B for steel bracing of mines.

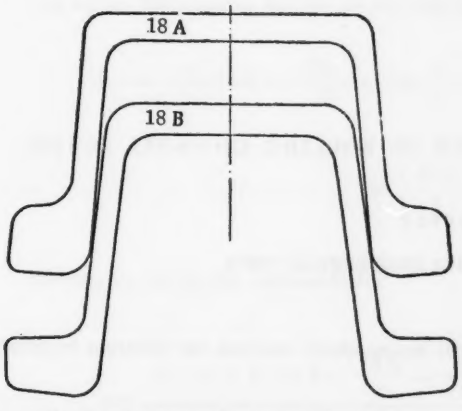


Fig. 2. Tightening of mine prop joint.

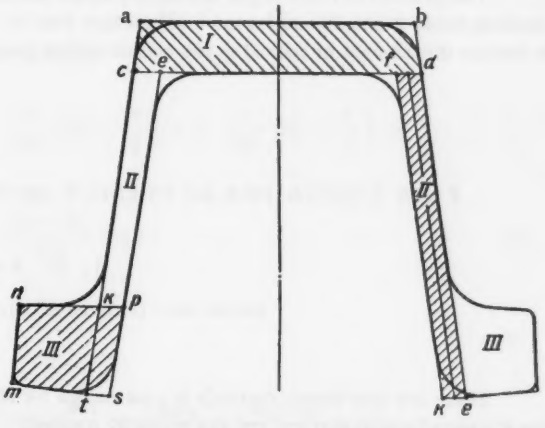


Fig. 3. Mine prop section divided into design parts: I) web; II) flange; III) rim.

Figure 3 represents the section, with the indication of separate design parts: web, flanges and rims. The dimensions of the rim are determined by the outline m-n-p-s. The formation of rims is bound mainly with the reduction in the thickness of the flange by means of some compression of the upper part of the section when it enters the next pass. At the same time, the upper part of the rim is drawn out about 0.5-1.5 mm. An additional factor affecting the formation of rims is their vertical compression with free lateral expansion in the open passes (Fig. 4). The side walls of the rims are formed by rolling once in the odd-number box passes. Apart from the rim formation, the compression of metal in the upper part of the pass is necessary also for the formation of the web, ab (Fig. 3). The maximum total lateral reduction in this direction is 50 mm for the 28B mine prop and 42.6 mm for the 28A prop. The reduction in props 18A–18B is smaller and is 28.5 and 15.5 mm respectively. This is connected mainly with the initial dimensions of the sections, the inclination of the flanges or the need for accomodating common passes in roughing rolls. In each individual case, this problem is solved differently but always as applicable to one or two conjugate sections.

The draft of the upper part of the section necessitates the application of upper compression of 3-5 mm.

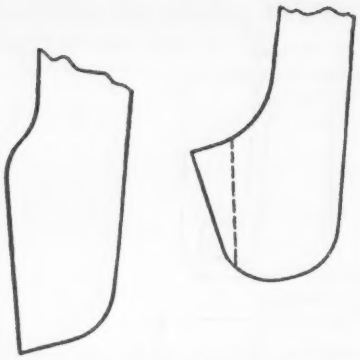


Fig. 4. Formation of the rims of mine props.

However, experience on rolling this section shows that it is more expedient to apply the same compression from below instead of from above. This is connected with the possibility of a change in the draft of thin webs when the piece could bend toward the bottom narrow guides and knock them out. Such incidents occurred mostly in the odd passes during a rolling of 18A–18B mine props. The lower compression has not such a great effect on the upper guides which are relatively wide.

In our case the upper compression in the channel-forming pass of the roughing stand should be not less than 100-115 mm. The taper of the outer walls of the flanges increases from the prefinishing to the first passes and reaches up to 20-30%. This was done in order to reduce the machining of rolls on dressing. In the first shaped passes of the roughing stand, however, the taper is reduced to 14-20% in order to ensure a steady position of the edge billet when it is the V-channel is cut in it.

Thus, in addition to deformation, during the rolling of mine props the bending of tapering flanges to the dimensions specified by GOST 5157-53 takes place. At the same time, the pass keeps the piece in alignment without any additional special attachments apart from the guides. The guides are adjusted in accordance with the width of the preceding pass and no compression is allowed. Otherwise the rolled piece becomes unstable and turns in the pass.

The method of roll design, developed at our Works for mine props and the successful production of these sections, made it possible to increase production year by year, to increase the output of defectless products and to reduce the amount of second grade and defective product.

PASS DESIGN FOR BUTTERFLY METHOD OF ROLLING CHANNEL BEAMS

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Senior Roll-Designer of the Salda Metallurgical Works

There are two known methods of pass design for channel beams, which methods are different in principle: the straight-flanged method and the butterfly method.

In practice, the straight-flanged method is most commonly used. The butterfly method is hardly used because of the existing view that the operation of the rolling mill becomes unstable owing to variations in the dimensions of the flanges of the section when this method of rolling is employed.

In 1954, the roll design for rolling channel beam No. 12 by the butterfly method was developed and tried out on the rolling mill of the Verkhne-Salda Works. A method of pass arrangement basically different from methods described in literature was applied in this design.

In literature on the rolling of the channel beam by the butterfly method, it is recommended to use passes in which the section is gradually bent, beginning from the first shaping pass to the finishing pass or in a few preparatory passes. In the method described here, the bending of the section takes place in the finishing pass alone (Fig. 1); in the preceding passes, beginning with the first in the rolling process and ending with the prefinishing, the section is shaped by the angle method without being bent.

The following defects became apparent in first tests of the new rolling method: variations in the thickness of the flanges and the underfilling of the external corners where the flange is joined to the web. These defects were eliminated by limiting the spread of the flanges in the finishing pass by means of closing the pass in that

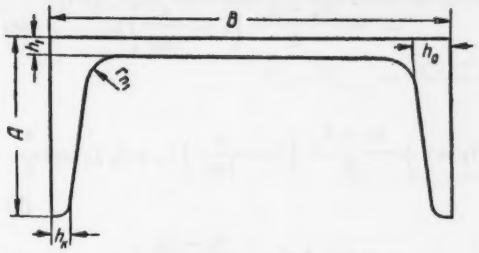


Fig. 1. Specified section.

part. The underfilling of the external corners was eliminated by adding false flanges in the prefinishing pass.

At the beginning of 1957, a butterfly-type finishing pass (Fig. 2) with a 10%, instead of a 1.5%, bending-out of the flange wall for rolling No. 12 channel beam was introduced. The section is straightened on a roller straightening machine.

Below, we give the design calculations of the passes for the butterfly method of rolling.

Finishing pass (Figs. 1 and 2). The principle of its design lies in the fact that the bending-out of the pass is achieved by bending the web. The dimensions of the finishing pass are made in accordance with the final specified section. Such a pass design is also used normally for ordinary straight-flanged finishing pass.

Pass No. 2 (Fig. 2b). For the center line of the flange we have:

$$C_1C_2C_3 = r\,\mathrm{tg}\,\frac{\alpha}{2} + \frac{\pi}{180}\left(90-\frac{\alpha}{2}\right)r + \frac{\pi}{180}\left(90-\frac{\alpha}{2}\right)\left(\frac{h_0\lambda_1}{2} - \frac{h_0-h_к}{360}\lambda_1\alpha\right).$$

The center line of the flange of No. 1 pass ($A-h_1+0.014\,A$) differs in length from the center line of the flange of No. 2 pass by the amount of its spread in No. 1 pass $\left(\frac{A\cdot h_1}{D}\right)$.

We set up the equation for the neutral line of the flange:

$$A-h_1+0{,}014\,A-\frac{Ah_1}{D} = r\,\mathrm{tg}\,\frac{\alpha}{2} + \frac{\pi}{180}\left(90-\frac{\alpha}{2}\right)\cdot r + \frac{\pi}{180}\left(90-\frac{\alpha}{2}\right)\times$$

$$\times\left(\frac{h_0\lambda_1}{2} - \frac{h_0-h_к}{360}\lambda_1\alpha\right). \qquad (1)$$

Hence, we obtain the expression for

$$r = \frac{A-h_1+0{,}014\,A-\frac{Ah_1}{D}-\frac{\pi}{180}\left(90-\frac{\alpha}{2}\right)\times\left(\frac{h_0\lambda_1}{2}-\frac{h_0-h_к}{360}\lambda_1\alpha\right)}{\left(\mathrm{tg}\,\frac{\alpha}{2}+\frac{\pi}{180}\right)\times\left(90-\frac{\alpha}{2}\right)}, \qquad (2)$$

where D – roll diameter.

λ_1 – deformation coefficient of the flange in No. 1 pass.

From Fig. 2b, we determine Φ_2, H_2, α, N_2, Δr, r_1

$$\Phi_2 = \frac{r}{\cos\frac{\alpha}{2}} + h_1\lambda_1\cdot\sin\frac{\alpha}{2}. \qquad (3)$$

The displacement of the curvature center O of radius r is:

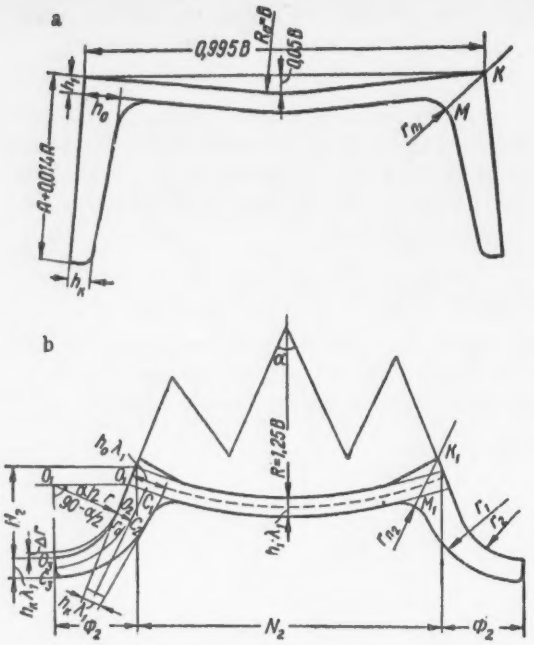


Fig. 2. Pass design: a) finishing, No. 1; b) No. 2.

$$\Delta r = \frac{h_0 - h_к}{2}\left(1 - \frac{\alpha}{180}\right)\lambda_1. \qquad (4)$$

$$H_2 = r + \frac{h_0 - h_к}{2}\left(1 - \frac{\alpha}{180}\right)\lambda_1 + h_1\lambda_1\cos\frac{\alpha}{2}. \qquad (5)$$

$$r_1 = r + h_0\lambda_1 - \frac{h_0 - h_к}{180}\lambda_1\alpha. \qquad (6)$$

For the web of the section, we assume R = 1.25 B. In the same way as for the flange, we set up the equation for the neutral line of the web:

$$B - 0{,}01\cdot B - 0{,}75T = \frac{\alpha\pi\left(R + \frac{h_1\lambda_1}{2}\right)}{180}, \qquad (7)$$

$$\alpha = \frac{180\,(B - 0{,}01B - 0{,}75T)}{\pi\left(R + \frac{h_1\lambda_1}{2}\right)}, \qquad (8)$$

where T – negative tolerance for the web length;
0.01 B – widening of the web in the finishing pass.

$$N_2 = 2R\sin\frac{\alpha}{2}. \qquad (9)$$

False flange. The height of the flange is calculated on the basis that the quantity $\frac{M_1K_1}{MK}$ is equal to or greater than the determined coefficient of deformation. In order to obtain the acute angle on the section, a slightly higher coefficient of deformation is taken.

The radius of the bend is:

$$r_{n2} \geqslant r_{n1}\cdot\lambda_1; \qquad (10)$$

the curvature radius of the flange-end rounding-off is:

$$r_к = (0{,}5 - 0{,}8)\,h_к\lambda_1. \qquad (11)$$

The radii of the bend and of the curvature of the flange end for subsequent passes are calculated in the same way.

Subsequent passes. To design n-th pass it is necessary to determine Φ_n, H_n, N_n and r_n (Fig. 2b).

The radius, r, is constant for all passes.

$$\Phi_n = \Phi_{n-1} - \left[\left(\frac{h_0 + h_к}{2}\right)\lambda_1\lambda_2\lambda_3\ldots\lambda_{n-1} - \left(\frac{h_0 + h_к}{2}\right)\lambda_1\lambda_2\lambda_3\ldots\lambda_{n-2}\right]k_n, \qquad (12)$$

where k_n – spread coefficient in the n-th pass

$\lambda_1, \lambda_2, \lambda_3 \ldots$ deformation coefficient in the 1st, 2nd, 3rd....n-th pass

$$H_n = H_{n-1} - \frac{h_0\lambda_1\lambda_2\lambda_3 \ldots \lambda_{n-1} - h_к \lambda_1\lambda_2\lambda_3 \ldots \lambda_{n-2}}{2}, \quad (13)$$

$$N_n = N_{n-1} - 0{,}005\, N_{n-1}\,(n-1), \quad (14)$$

$$r_n = r_{n-1} + \frac{h_0 + h_к}{2}\,\lambda_1\lambda_2\lambda_3 \ldots \lambda_{n-1}. \quad (15)$$

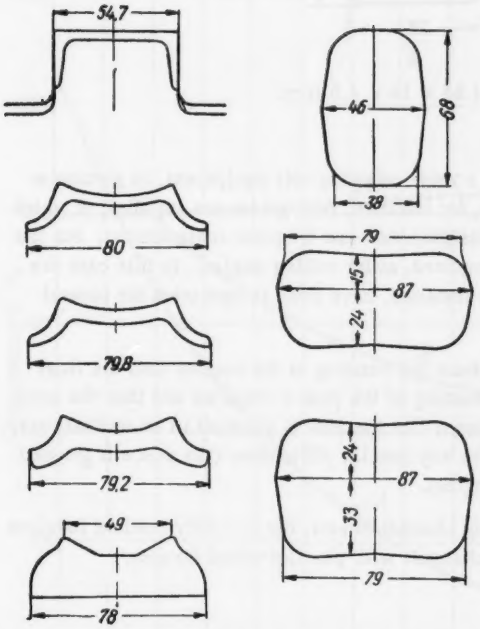


Fig. 3. Roll passes for rolling window-frame Section No. 8 (55×25×4 mm).

The butterfly method of rolling channel beam was applied also for rolling window-frame section No. 8 (55 × 25 × 4 mm); Fig. 3) and channel (26 × 14 × 4.5; Fig. 4).

The butterfly method of rolling steel permits larger drafts and, hence, it is possible to reduce the number of passes. In the rolling of channel beam No. 12, two passes were eliminated, and this resulted in an increase in the temperature of steel at the end of rolling and a reduction in roll consumption, particularly of finishing rolls. All this increased the mill output.

The main advantage of the butterfly method over other methods is a substantial economy in roll consumption owing to a smaller amount of machining when the rolls are dressed. For instance, in the course of dressing the passes, 25-30 mm is machined off the diameter of the rolls with straight-flanged passes, but only 5-6 mm if the butterfly method of rolling is employed. At the same time, with the butterfly method of rolling, 2 or 3 times more steel can be rolled for one change of rolls than with the straight-flanged method.

The application of the butterfly-type finishing pass increases its life considerably and reduces by half the amount of metal machined off during roll dressing. In the butterfly method, the roll passes are fully restored by dressing and, therefore, there is no need to replace finishing, intermediate and roughing rolls depending on the lateral wear of passes.

The groove depth in the rolls for the butterfly method of rolling is approximately half as great as for other methods and, hence, it is possible to increase the draft in the passes without risking a fracture of the rolls.

Some authors assume that the butterfly method of rolling channel beam requires a longer roll barrel (body). Experience on rolling No. 12 channel by this method on the mill at our Works showed the opposite; the roll-barrel length necessary for accommodating the passes is substantially reduced on account of the reduction in roll collars and in the number of passes.

The butterfly method of rolling channel beams fully satisfies GOST specifications. The fear that owing to variations in flange thickness a high proportion of second-grade product would be obtained were not justified. The application of the butterfly-type finishing pass for No. 12 channel improved the surface quality of the product. During the rolling of No. 12 channel from 240 mm square ingot by the butterfly method over a long period of time, no cracks or seams on the flange edges were detected.

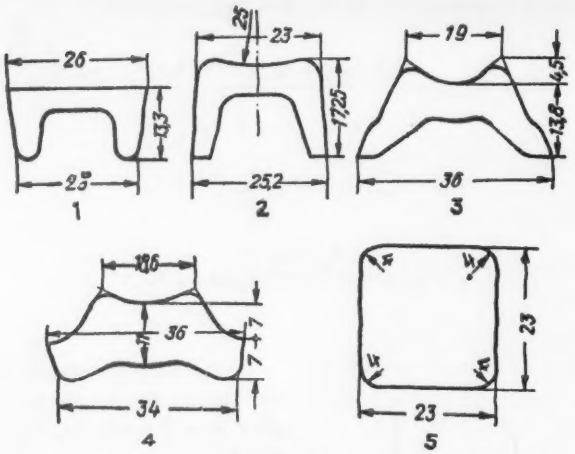


Fig. 4. Roll passes for rolling channel 25 × 14 × 4.5 mm: 1)-5) passes.

For rolling the channels by the straight-flanged method a more complex roll equipment, in particular a large number of guides, is required. In the prefinishing pass, for instance, four guides are installed of which two flange-guides are worn down rapidly and require careful maintenance and frequent replacement. For the butterfly method of rolling, a simple auxiliary equipment is required, as for rolling angles. In that case the life of auxiliary equipment is relatively long. The guides, for instance, have been in operation for several months without replacements.

At first sight, it might seem that in the finishing pass, where the bending of the section into the final shape takes place, an especially careful screwing down and adjusting of the pass is required and that the entry of the piece into the pass is rather difficult. Experience has shown that the pass is adjusted in an ordinary way, as for rolling angles, and the entry of the piece does not involve any special difficulties (the piece is gripped by the rolls somewhat easier even than during the rolling of angles).

In comparison with the straight-flanged method of rolling channel beams, the butterfly method involves no particular difficulties in rolling light-section channel and channels with parallel-sided flanges.

CAUSES OF CORNER CRACKS ON BLOOMS FROM RIMMED-STEEL INGOTS

S. V. Makaev

Chief Engineer of the Nizhne-Tagil Metallurgical Combine

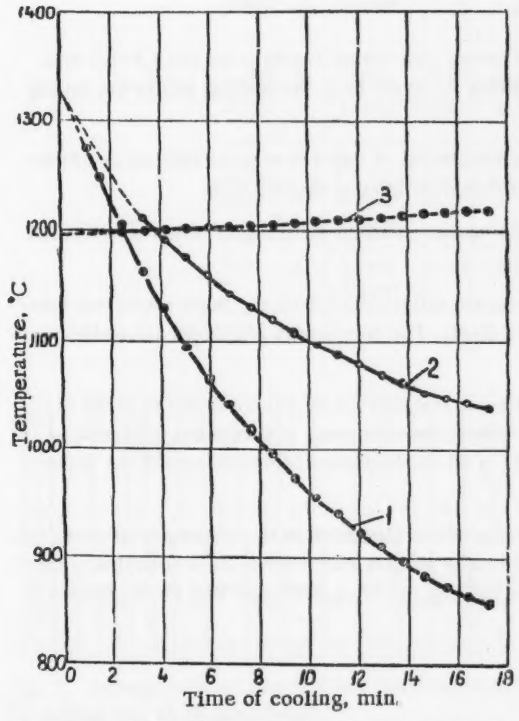
When blooms and slabs are rolled from large ingots, in particular rimmed-steel ingots, there often appear, on the corners of the rolled piece, cracks which are removed by surface conditioning if they are not too deep. This defect often is the cause of rejecting the steel product.

For combatting the defectiveness of blooms and slabs, it is very important to determine the causes of crack formation. One of the very important factors which promote this defect is nonuniform temperature over the cross section of the ingot, this nonuniformity resulting from the cooling of the corners if there is a delay before the ingot is rolled. The following test was carried out in order to confirm this.

Three holes of 35 mm diameter each were drilled half way up a 7-ton ingot, in such a way that one could insert into them thermocouples for measuring the temperature in the center of the ingot, in the corner

Results of the Inspection of 250 x 250 mm Billets

Time of ingot cooling	Number of billets inspected	total		Affected by corner cracks							
				one edge		two edges		three edges		four edges	
		number	%	number	%	number	%	number	%	number	%
1-1.5	166	26	15.6	15	9	8	4.8	2	1.2	1	0.6
2.6-4.0	176	55	31	22	12.5	22	12.5	9	5.1	2	1.1
5.0-7.0	149	93	62	10	6.6	17	11.3	30	20	36	24



Cooling of rimmed-steel ingot. 1) Corner (edge) of ingot; 2) side; 3) center.

and in the middle of the side wall 10 mm below the surface. The ingot was heated together with other ingots in the blooming-mill soaking pit to a temperature of 1320°C in 7 hours in accordance with the existing instructions. After the ingot was discharged from the pit, the thermocouples, which had been connected to high-speed recording potentiometers EPP-120 were inserted. To eliminate time losses on heating the hot junction of the thermocouples, the junctions were heated beforehand to 1100-1180°C.

The time interval between the delivery of the ingot from the furnace and the moment when the setting up of the thermocouples was completed constituted 1-1.25 min. The graph shows the temperature of the corner (edge), side and center of the ingot.

It is seen from the graph that at the moment when the ingot is delivered from the furnace the temperature at the center of the ingot is 125° lower than at the surface since the ingot was cold when charged. An accelerated cooling of ingot corners takes place.

In 2.5 min, the temperature difference between the center and the corners disappears, in 4 min it reaches 75°C and in 7 min it is 170°C.

It is absolutely obvious that for ingots which are charged hot into the soaking pit and whose temperature at the center is 1250-1300°C, the temperature difference between the center and the corners increases by 100°C and after 2.5 min it is more than 100°C and after 7 min it constitutes more than 250°C.

If we take into account that for a 100° temperature change (in 1000-1250° temperature range) the resistance to deformation changes by a factor of 1.5, it becomes obvious that the nonuniformity in deformation (tensile stresses in ingot edges) increases as the time of cooling before rolling increases.

To verify the effect of the nonuniform deformation, caused by the low temperature of the ingot corners, on crack formation at bloom corners, tests were carried out in the course of rolling the ingots which were cooled for 1 to 7 min in the period from the moment of discharge from the pits to the moment of entry between the rolls.

Seven-ton rimmed-steel ingots, cast from above into square molds were rolled to 250 mm square billets in 13 passes in accordance with the scheme of drafts as accepted at our Combine. After they were rolled, the billets were cut longitudinally into three parts. When cold, they were inspected. The results of the inspection are given in the table.

Ingots of 3kp steel from one heat were rolled at the blooming mill into 300 × 320 mm billets and without any intermediate conditioning were reheated in the furnaces of the rail-structural mill and rolled into 125 mm squares.

Before being rolled at the blooming mill the ingots were kept for 1 to 4.5 min. The results of the inspection of billets from these ingots are shown below:

	Time of ingot cooling, min		
	1-1.5	1.6-2.5	2.6-4.5
Number of billets inspected	57	74	50
Billets affected by corner cracks			
Number	3	30	28
%	5	40	56

It is seen from the table that the minimum per cent of billets with corner cracks is for the 1.0-1.5 min cooling time of ingots and this is the normal time for transporting the ingot from the soaking pits to the rolling mill.

With an increase in this time to 5-7 min, the effect of the cooling of edges is so great that crack formation takes place on all four edges (corners) and the amount of defective blooms exceeds 60%.

To investigate the character of cracks appearing on blooms from rimmed-steel ingots which were cooled before rolling, samples were taken for metallographic study.

The macrostructure of the specimen cut from 250 mm square billet rolled from the ingot which was cooling for 5-6 min represents a normal macrostructure of rimmed steel. The honeycomb blowholes are welded. The maximum depth of surface fractures is 7-10 mm.

The investigation of the defect under a microscope indicates that there is no oxidation at the place of rupture. As a rule, the ruptures are blunt. The characteristic symptoms connected with ruptures occurring at the place of nonwelded blowholes have not been found. There is no accumulation of oxides around the defective spot.

It follows from the investigations carried out that the cause of the disruption in the continuity of steel, i.e., the cracks on the corners of the rolled piece, are longitudinal tensile stresses which result from nonuniform deformation; and the longer the time of cooling the ingot before rolling, the more extensive will be the formation of corner cracks on the billets.

REDUCING THE CONSUMPTION OF MILL ROLLS

B. M. Iliukovich, Senior Roll Designer at the Chusov Metallurgical Works and

A. S. Bulgakov, Senior Foreman at the Salda Metallurgical Works

Worn rolls, which previously were written off as rejects, are used on the three-stand one-line mill of the Chusov Metallurgical Works.

The first tests on the use of such rolls were carried out during the rolling of channel beams. The positioning of the passes on the rolls of the newly prepared set of the finishing stand for rolling No. 10 channel is shown in Fig. 1a. The difference in rolling diameters at the channel wall (base) constitutes 30 mm. In the course of dressings, this difference may be reduced to 15 mm on account of less extensive machining of the lower roll. The resulting displacement of the neutral line affects neither the quality of the section rolled nor the life of the passes.

Operating Statistics of the Rolls of the Finishing Stand for Rolling No. 10 Channel Beam

Roll No.	Shore hardness	Position of the roll on the stand	Chemical composition of rolls, %							Total time of operation, hr	Quantity of steel rolled, tons
			C	Si	Mn	P	S	Cr	Ni		
Newly-made rolls											
3239	47	Upper	3,09	1,62	0,71	0,31	0,008	0,6	1,07	152	2297
25182	45	Lower	3,27	1,6	0,67	0,21	0,005	0,54	0,99		
755	45	Upper	2,98	1,05	0,67	0,25	0,006	0,66	0,89	176	2826
25187	49	Lower	3,00	1,51	0,55	0,29	0,003	0,70	0,98		
Twice-used rolls											
3939	47	Lower	3,09	1,62	0,71	0,31	0,008	0,6	1,07	144	2613
		Upper								160	2921
3937	47	Lower	3,03	1,49	0,56	0,30	0,007	0,52	0,80	200	3416
		Upper								182	3120

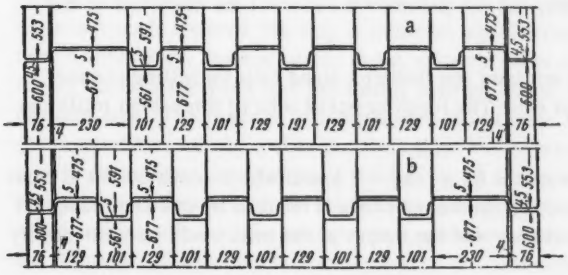


Fig. 1. Arrangement of passes of the finishing-stand rolls for rolling No. 10 channel beam.
a) Before the lower roll is worn down; b) after the lower roll is dressed and used as the upper roll.

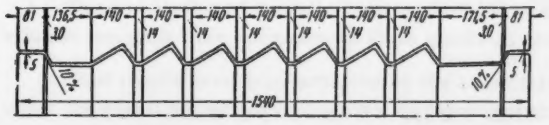


Fig. 2. Arrangement of the passes of the finishing-stand rolls for rolling an angle.

In the course of operation, the rolls are dressed and their diameter is reduced by not more than 50 mm as compared with the design diameter. In view, however, of the very slight taper of the outer sides of the flanges, the side wear of the passes is, as a rule, the cause of rolls going out of commission, except for the finishing stand where the flanges have a larger taper (5-10%).

After the rolls become unusable for further work (roll fractures are not taken into account), the upper roll is written off as reject and the lower is dressed and mounted as the upper roll into a newly prepared set (Fig. 1b). The passes on the rolls are slightly displaced to the left and the grooves of the upper roll correspond then to the collars of the lower roll, which can be dressed (remachined).

The main condition for the feasibility of dressing the lower rolls is a preset positioning of grooves and collars on the roll barrel.

The width of the intermediate and the outer collars is chosen in such a way that after the dressing of the lower roll a completely new set of passes, which exclude any change in the method of rolling, is obtained. The difference between the diameters of the outer collars is taken as equal or slightly larger than the maximum reduction in the roll diameters in the course of the rolling process.

In the table, the data are given on the durability of rolls of the finishing stand for rolling No. 10 channel:

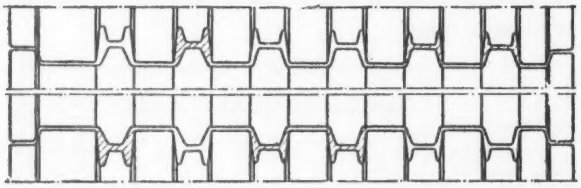


Fig. 3. Arrangement of the passes of the roughing-stand rolls for rolling No. 10 H-beam.

of newly made set and of rolls (No. 3939 and 3937) which were used as bottom rolls and then, after having been dressed, as upper rolls.

When rolls made of low-alloy magnesium-modified cast iron are used twice, no lowering in roll life is observed.

The upper roll for rolling 90 × 60 × 6–8–10 and 80 × 55 × 6–8–10 mm angles is also used twice (Fig. 2) but after it is dressed and adjusted as the upper roll, a roll set is obtained which contains eight instead of seven passes.

Figure 3 shows the arrangement of the passes of the roughing-stand rolls for rolling No. 10 H-beam. The set consists of four rolls. After the wear of the passes exceeds the permissible limit, the rolls have to be changed; the upper and lower rolls are interchanged and a new middle roll is fitted. The arrangement of the passes on the roll barrel and the width of the collars are chosen in such a way that, when a set of rolls is out of commission on reaching the minimum limiting diameter (the width of the passes is restored by dressing), the lower and upper rolls can be made into two middle rolls for a new four-roll set. At the same time, the grooves of the middle roll correspond to the collars of the lower and upper rolls, and the passes in the new set are displaced to the left.

The double use of rolls is practicable for various sections: the finishing-stand rolls for rolling channel beams, H-beams, angles, channels with bent web, hoops etc. The roughing-stand rolls of three-high mills can also be remachined and used twice (see Fig. 3).

As a rule, the double use of the roll is more economical for a long roll barrel which accomodates at least five or six passes, because when the rolls are remachined the number of passes is reduced by one and, hence, if the roll is short, very frequent roll changes would be necessary and the output of the mill would be substantially lowered.

At the mills which operate on the principle of one pass in each stand, it is possible to remachine the rolls of the roughing stands too.

In general, the problem of using the rolls for a second time should be solved in relation to the operating conditions of each mill; there is, however, no doubt about the expediency of this method.

When rolls are used twice, their consumption is markedly reduced, the life of passes remaining the same.

AUTOMATION OF TUBE-CUTTING AND TUBE-THREADING MACHINES

Eng. T. I. Romersntein

A large number of tube-cutting and tube-threading machines are in use at tube-industry works. At some works, the operation of these machines was partly mechanized during postwar years. Until recently, however, no automation was applied to these machines. Therefore, the automation of a few groups of tube-cutting

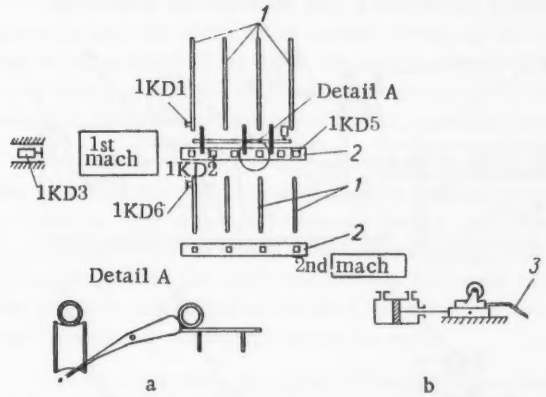


Fig. 1. Automation scheme for the tube-cutting machines: a) batching device; b) mechanism for the removal of burrs, 1) racks; 2) conveyer; 3) tool.

(type 9A151) and tube-threading machines which was successfully introduced at the Lenin Dnepropetrovsk Works by the laboratory of automation mechanization of the tube industry, is of great interest.

Figure 1 shows the layout of the contact pickups of a pair of cutting-off machines. The automatic operation of the machines is as follows: a set of tubes whose ends are to be cut off are placed on the racks of the first machine by means of a bridge crane. The tubes roll off to the batching device and come to rest in the position shown in Fig. 1 (detail A). The automatic scheme operates at the moment when the tube closes the 1KD1 pickup, which serves for checking if the tube is in the batching device which throws off the tube onto the conveyer of the first machine. When it rolls down the batching device fingers, the tube closes the 1KD2 pickup which switches the conveyer drive to move the tube "forward."

When it comes into the extreme forward position, the tube touches the stop with its front end and, at the same time, closes the 1KD3 pickup which is mounted in the stop and which, after a suitable lapse of time, switches on the hydraulic mechanism for gripping the pipe in the chuck. The time lapse is necessary to stop the vibrations of the tube on the conveyer caused by the buffer action of the stop. After the pressure (controlled by the pressure relay of the machine) becomes established in the hydraulic system, the pulse is given for the forward movement (i.e., towards the tube) of the carriage saddle with the tools, the conveyer rollers are stopped, the conveyer is lowered, the stop is retracted and the cutting of the tube end begins. When the cutting operation is completed, the end switch at the extreme forward position of the carriage saddle gives the signal for taking the carriage back into the initial position and to switch in the mechanism for the removal of burrs from the tube end. The removal of burrs is carried out after a suitable lapse of time determined by the time relay of the machine. At the end of the time lapse the conveyer rolls of the first machine are started to move the tube back and the conveyer itself is lifted up to the upper position. After the carriage has reached the initial position and the stop has been set in the operating position, the tube is released; it moves on the conveyer backwards and when it is in the extreme rear position it closes the 1KD5 pickup which gives the signal for starting the throw-out device which in turn transfers the tube onto the racks of the second machine which operates similarly to the first.

The mechanism for the removal of burrs from the tube end consists of a pneumatic cylinder with a piston whose rod is hinge-joined to a slide block. On the slide block is mounted a roller to whose base is attached a tool holder with a tool. The roller is in contact with a curvilinear guide. The piston actuates the slide block and the roller. The tool holder with the tool describes a curvilinear path together with the roller and approaches the end of the tube under treatment.

The automation of the tube-cutting machines makes it possible for one cutting tool to serve two machines, whereas two tools were necessary for these machines previously.

Good results have been achieved by the automation of the tube-threading machines of "Stamets" type; under the new conditions two machines require only one threading die. The operation of the automated tube-threading machines (Fig. 2) consists of the following.

Tubes, in batches, are loaded by a bridge crane onto the racks of the first machine of a given pair of machines, and then roll down the inclined racks until they are stopped by the projections of the lifting table which contains a conveyer. The automatic operation of the machines is started by pressing a button for lowering the table. One tube rolls down onto the conveyer which is automatically set for "forward motion" and inserts the tube into the chuck. On moving forward, the tube end touches the KD stop which gives the signal for starting the gripping mechanism. After a time lapse (about 0.5 sec), the motor which moves the carriage

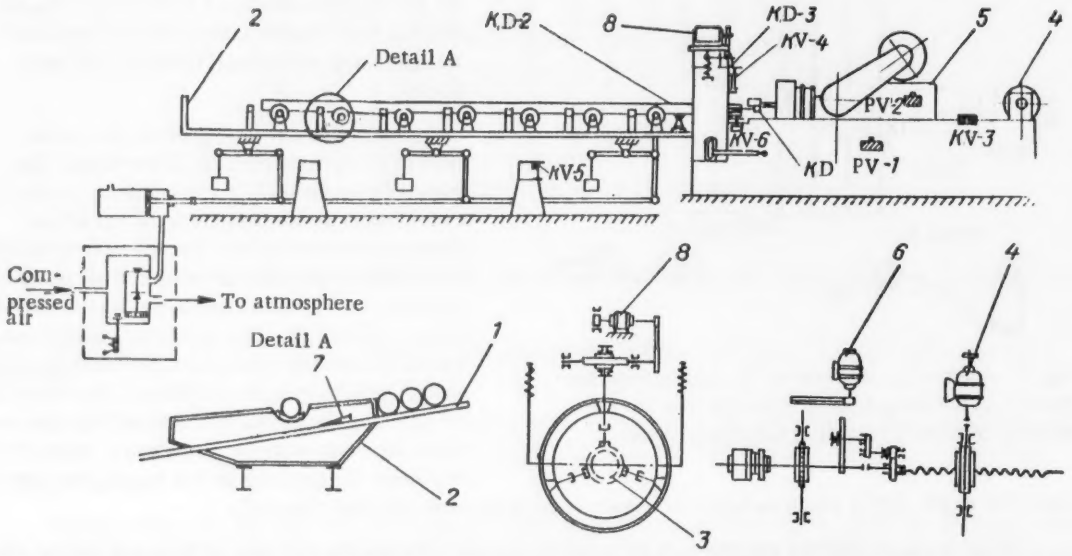


Fig. 2. Automation scheme for the tube-threading machine: 1) inclined racks; 2) lifting table and conveyer; 3) chuck; 4) electric motor driving the carriage; 5) carriage; 6) electric motor driving the spindle; 7) stop; 8) electric motor driving the gripping mechanism.

forward is switched in. In its forward movement, the carriage presses the PV-1 switch which gives the signal for changing the fast movement of the carriage to the automatic gear and for starting the spindle.

When the cutting of the thread is finished, the tube presses against the sprockets of the threading die which opens and, by its movement, resets the PV-2 switch which causes the carriage to start moving back. The carriage is stopped in its rear position by the KV-3 end switch. Simultaneously with the back movement of the carriage, the signal for opening the chuck and moving the conveyer in the backward direction is given. The opening of the chuck is discontinued when the KV-4 end switch operates. To speed up the disconnecting of the releasing mechanism, a KD-3 contact pickup is also installed.

As soon as the tube gets out of the chuck and outside the KD-2 brush pickup the signal for lowering the table is given. The tube rolls down the inclined grid to the second machine.

On descending, the table with the conveyer presses the end switch KV-5 and then begins to rise under the action of a weight or a pneumatic drive. At the same time, the order is given for the forward movement of the conveyer. The whole process is repeated in the same sequence. The operation of the second machine of the given pair is arranged in the same way. Both machines can be started and stopped from one and the same control desk.

NEW ROLLING SCHEME AT THE BLOOMING MILL

V. D. Zotov

Senior Operator at the Blooming Mill of the "Krasnyi Oktiabr'" Works

By adopting a new rolling scheme, the shift to which I belong has increased output by 8% since 1955.

All work at the blooming mill is carried out simultaneously by three operators. The supervision rests with the senior operator whose duties include: setting up the upper roll and the draft of the ingot (in 100 sec), checking the temperature of the ingot, the scheme of ingot drafts, the correct dimensions of the assigned section and the reversing gear of the machine. The operator of the manipulators needs only 25 sec to turn the ingot and transfer it from one pass to another. Thus, the senior operator has four times as much work as the manipulator.

Much time was lost at our mill (during the rolling of 6-ton ingots) after the last run of the piece in the first pass when the piece was discharged in front of the mill and, at the same time, the upper roll was raised to allow the entry of the piece into the second pass. After turning the piece, the manipulator was also waiting for the upper roll to be lifted. The mill was idling for 3 min during the rolling of every alternate ingot. On my suggestion, the last run in the first pass was made idle. In order not to lower the output, the drafts in the first pass were rearranged and slightly increased. In this way, it was made possible to finish the reduction of the ingot when it was discharged behind the mill.

At the same time, the upper roll begins to be raised and the ingot is discharged in front of the mill.

By the time the upper roll is raised, the manipulator has turned the ingot and is entering it into the second pass. In this way, we eliminated the idling of the mill before the entry of the ingot into the second pass, and increased the output by 2.5%.

In addition, with a view to increasing output, we reduced the number of runs in the first pass from 4 to 2 when rolling 5-ton ingot, and began to roll 4-ton ingot without turning it after the second pass.

There still remain many hidden potentialities for output increase and their utilization will markedly raise operating efficiency.

AUTOMATIC TONGS FOR GRIPPING STEEL

B. Ia. Grigor'ev

Senior Foreman at the Soaking Pits of the Heavy-Section Mill of the A. K. Serov Metallurgical Combine

A new design of tongs for gripping steel (Fig. 1) were made at our mill. At present, two out of the three cranes above the soaking pits are equipped with such tongs. The tongs have been in operation for three years and have proved to be highly efficient in operation. Their design and the principle of operation are as follows: the tongs consist of a cast steel frame with slots in which are inserted jaws mounted on hinge pins in a connecting block. A catch is fitted on a hinge pin in the upper part of one of the jaws. The tongs are suspended on the hook of the pulley system of the crane by means of a shackle which is joined to the upper part of the frame by means of a pin with a bush.

In the outer vertical slots of the frame, forged guiding plates, rigidly joined to the connecting block by rivets and electric welding, are accommodated. The guiding plates keep the jaws on the same level all the time. The pin (4) (Fig. 1) has a round head on one end and a rectangular hole on the other; a key is inserted into the hole. The jaws have inclined slots which slide on the pin and bush when the frame is lifted or lowered. In the upper part of the frame, there are pins which make the jaws always come to rest in a definite position in the upper part, so that the necessary distance between them for locking the catch is maintained.

In Fig. 1a, the tongs are shown in the open position in which they are firmly kept by the catch. In this position, the tongs are lowered onto the ingot or the lid of the soaking pit. On descending, the finger (14) with the bush comes against the rear end of the catch and turns it into a vertical position. This position is shown in Fig. 1b.

The gripping of the ingot is effected by lifting the hook and with it the frame of the tongs. At the same time, the pin (12), by getting into the inclined section of the slots of the jaws, forces the lower ends of the jaws

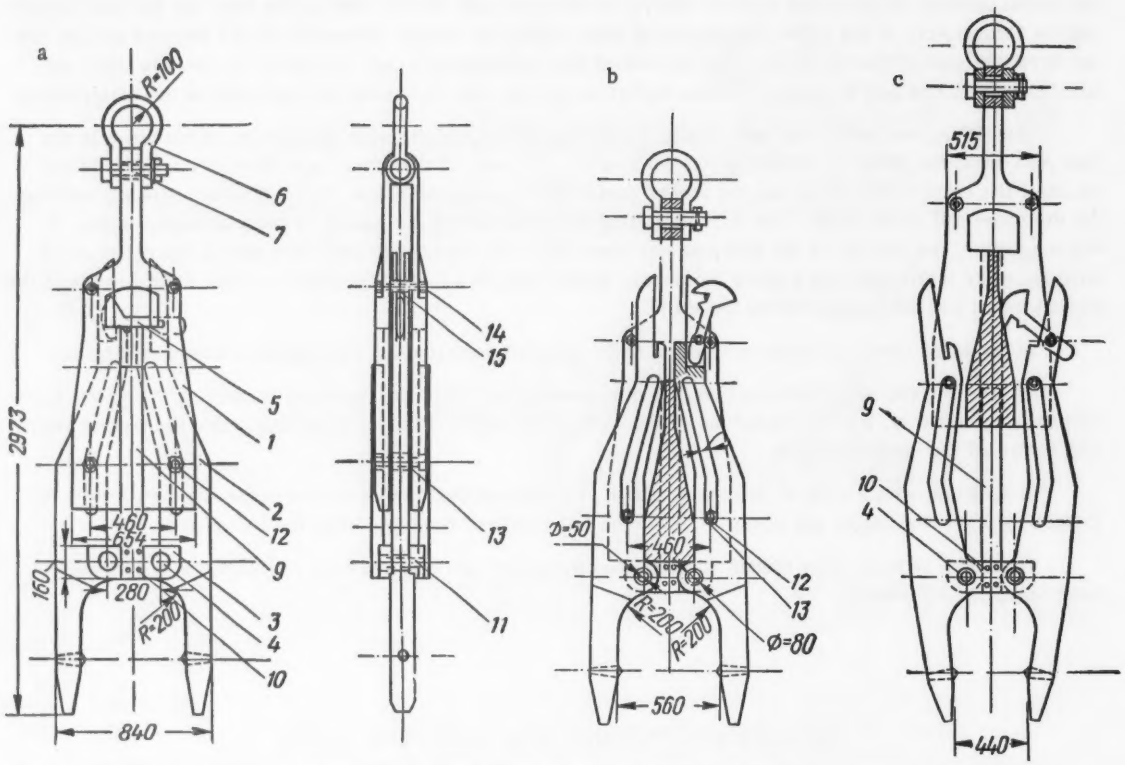


Fig. 1. New-design tongs: a) when open; b) before gripping the ingot; c) when fully closed; 1) frame; 2) jaw; 3) connecting block; 4) pin; 5) catch; 6) shackle; 7) 8) 12) 14) pins; 9) guiding plates; 10) rivets; 11) key; 13) 15) bushes.

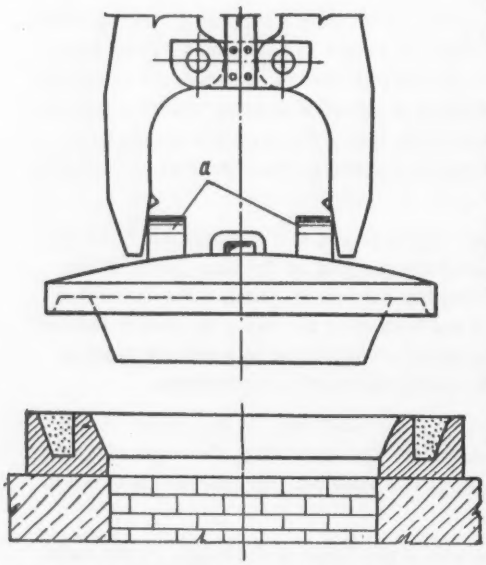


Fig. 2. Tongs gripping the lid: a) lugs.

to close and to grip the ingot. The pin (14) throws the catch back into a horizontal position, but with some delay so that by then the jaws have time to open their upper ends.

The maximum and minimum span of the tongs was selected to suit the dimensions of the cross section of ingots produced at the Combine. In the given case, the difference between the maximum and the minimum span is 120 mm. In addition, an adjustment on hinges of about 100 mm is possible. If necessary, the maximum and minimum opening of the tongs can be changed by changing the inclination angle or the length of the slot in the jaw.

In order that the tongs could grip the lid, it was provided with two lugs (Fig. 2). The frame of the tongs is cast (steel 35); the life of the jaws (forged from steel 10-25 and then machined) until they are completely worn is 4-8 months. Cast jaws are being prepared for testing. Pins and bushes are made of steel 20Kh2N4A and are subjected to thermal treatment in accordance with GOST. The life of the tongs until they are completely worn is 6-12 months or even more. The guiding plates (forged from steel 10-35) can be used without being machined.

With the introduction of the new-design tongs the work of

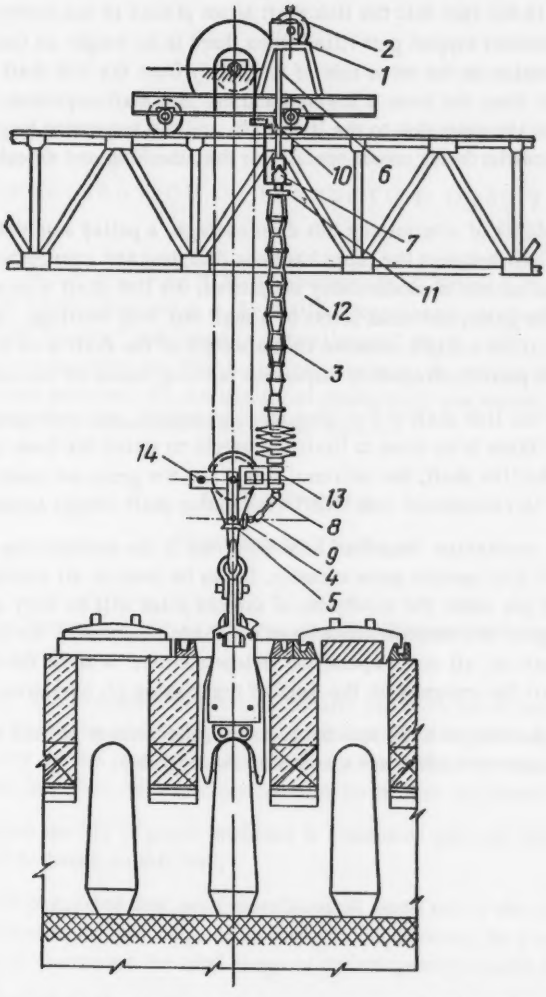


Fig. 3. Diagram of the mechanism for rotating the tongs: 1) worm gear; 2) electric motor; 3) link shaft; 4) 5) bevel gears; 6) reduction gear shaft; 7) 8) 9) universal joints; 10) bushing; 11) shackle; 12) rope; 13) pin housing; 14) counterweight.

welders was made much easier, and the lifting of ingots from ingot cars, the charging of ingots into the soaking pits, the removal and replacement of lids was made much speedier so that it was possible to make 16 men available for other duties. Initially the tongs, however, had to rotated by hand. Now, a turning mechanism (Fig. 3) has been designed, made and tested; it consists of a reduction worm gear with an electric motor mounted on the crane bogies, a link shaft and two bevel gears. The button control of the electric motor is in the crane operator's cabin. The electric motor, of 3.6 kw and 720 rpm, rotates the reduction gear shaft which is connected by an universal joint to the link shaft. By means of universal joints and bevel gears, the hook and, with it, the tongs are rotated.

For safety reasons, the link shaft is supported by a shackle and rope passing on both sides along the hinges of the shaft. In its lower part, the link shaft ends in a pin mounted vertically in a pin housing with two roller bearings. A counterweight is attached to the pulley support on the opposite side from the link shaft in order to reduce the tilting of the suspension support by the link shaft.

When the above experimental model of the turning mechanism was put into operation, a basic defect in design was detected; it lies in the fact that the link shaft is not placed in the center line of the pulley suspension and, therefore, the suspension support gets tilted when there is no weight on the hook. The counterweight, mounted on the pulley suspension on the other side of the pulley from the link shaft counterbalances only the lower part of the mechanism when the hook is lowered and the link shaft expanded. As the hook is raised the load on the pulley suspension increases due to the link shaft, and the suspension begins to tilt and bends the link shaft. This causes difficulties during repair operations; the attaching and detaching of the link shaft takes about 30 minutes.

At present, with the object of eliminating this disadvantage, a pulley suspension in which the link shaft is positioned on its vertical axis between the ropes has been designed and constructed; at the same time, the lower part of the turning mechanism is considerably simplified, the link shaft is attached directly to the hook screw and there is no need for gears, universal joints or a shaft box with bearings. In addition, the plates of the link shaft are made thicker, since a slight increase in the weight of the shaft is of no significance when the shaft is placed centrally. At present, the pulley suspension is being tested on two cranes.

The first specimen of the link shaft is 7 m long when expanded, and contracts to 700 mm when the hook is in its top position, so that there is no need to limit the height to which the hook can be raised. To make the whole mechanism lighter, the link shaft, the universal joint, and the gears are made of 20Kh2N4A steel and subjected to heat treatment in accordance with GOST (the pinion shaft weighs approximately 100 kg).

Results of tests on the mechanism described here show that if the shortcoming which was pointed out above is eliminated the mechanism can operate quite steadily, it can be used on all cranes operating above soaking pits and so the operations at pits under the conditions of the old plant will be fully mechanized. Additional controls in the operators cage of the electric crane have not made the work of the operator difficult, since they have made it possible to carry out all crane operations independently. Without the mechanization, the crane operator had to coordinate all his actions with the work of the soaking pit operators.

When the turning mechanism with the new tongs is used, the amount of work for the soaking pit operators is reduced by half and the crane operations are speeded up considerably.

ON THE ORGANIZATION OF PRODUCTION QUALITY CONTROL

In the article by N. P. Inozemtsev, Ia. I. Sokol, I. F. Rysev, D. A. Tarasenkov and S. I. Zamiatin "Organization of Production Quality Control" published in "Metallurgist" No. 9, 1957, the problem of reducing the OTK staff and increasing the responsibilities of plant personnel for the quality of production, was raised. We publish here some comments on this article.

G. D. Rogoza

"Zaporozhstal" Works

The problem, raised in "Metallurgist," of the organization of inspection control at metallurgical works is extremely important and timely.

General opinion, which was expressed in previous articles and with which one should agree, is that the number of OTK personnel is excessive and that it should be reduced. It is generally agreed, too, that it is necessary to place the responsibility for the quality of production on the works foremen, section and plant heads. As regards some other problems, however, contradictory opinions have been expressed.

To ensure an appropriate quality of goods produced is a national task and the organization of inspection control for this purpose must be based on this task.

Therefore, it cannot be permitted that each metallurgical works solves this problem in "its own way" and has different organizational forms and range of control. In our opinion, for a correct organization of technical control it is necessary to determine the total range of the obligatory control of:

1) raw materials, received by the works;

2) production operations

3) semiproducts

4) finished products

The securing of the obligatory norms of the qualitative characteristics given by All-Union standard specifications should be determined by the obligatory range of the control operations on the production process which determines the quality of the metal. One should not forget that the control tests do not always characterize the whole heat as opposed to the correctly established and maintained operating conditions which, therefore, should be the subject of an objective control.

It is impossible to explain a large number of deviations from established operating conditions simply by the "carelessness" of the plant personnel. Moreover, we think that a substantial number of failures in maintaining the operating conditions (and especially in steel production) are due to several causes which cannot be remedied by the respective operators. Nevertheless, for the correct designation of the metal, for learning from experience, for improving operating conditions and the quality of the product (in particular of special-duties steel), a systematic, objective and rigorous control of the most important technological operations and an appropriate recording of the control results are necessary.

Plant or section	Materials and technological operations subject to control	Who carries out the control	Remarks
Internal-acceptance section	Quality and acceptance of raw materials	OTK	Necessary only at works with the full metallurgical cycle
Sinter plant	Quality of the charge Sintering process Quality of the sinter	Production personnel	–
Blast-furnace plant	Acceptance of raw materials at the ore yard Blending of ores Charging the materials into bins Charge determination and the manner of charging the blast furnace Scale-car operation Technology of pig-iron smelting Delivery of charge materials to the furnace top Operation of the charging equipment	Production personnel	–
	Quality of pig iron	OTK	
Open-hearth and electric-steel-making plants	Materials in the stock yard	Production personnel	
	Melting and pouring	OTK	
	Quality of the preparation of molds	Production personelnel	OTK, at electric steel-making works
	Marking and designation of the heat	OTK	
Rolling mills	Charging and discharging the ingots at the heating furnaces	OTK	
	Technology of rolling process	Production personelnel (foreman)	
	Taking samples for tests in the course of the heat and for final tests	OTK	–
	Grading and acceptance of rolled product	OTK	
	Loading the product and making out the certificates	OTK	
Heat-treatment plants	Technology of heat treatment	Production personnel	
	Taking samples for tests	OTK	–
	Acceptance of finished products	OTK	
Test laboratories	Control tests in the course of the heat and final control test, and all types of tests (except chemical analysis)	OTK	–

The steelmaking process constitutes a decisive link in metallurgical production, and the technology of the process is already adequately developed. The maintenance of the established parameters of this process depends on technological instructions; violation of the instructions is often the cause of defects in product detected only in the course of use by the consumer.

There is no guarantee that deviations from normal technological process will be shown in the reacords maintained by the production personnel if an objective control by the OTK is absent. After all, the performance of the production personnel should be and is judged by the results in fulfilling the production plan and this factor is dominant in the work of production personnel. Very often, however, this factor can have a negative effect on the quality. At the same time, there may be cases when failures in maintaining operating conditions will not manifest themselves immediately, either in the results of the operation or in the qualitative characteristics determined in tests in the course of the heat and final tests, but will be revealed only when the final product is being used by the consumer.

In our opinion, the transfer of the control functions to the production personnel can be and should be carried out only on those operations where any irregularities will have immediate effect on the normal performance of the unit (defective products, nonfulfillment of the plan, etc.). To those operations belong all the technological operations of sinter production, blast-furnace production, the control of the preparation of molds for pouring ordinary-quality steel, the temperature of ingots before rolling, the regime of thermal operations, etc. In addition, in all cases where the control of the operating conditions effected by recording instruments, there is no need for constant control by OTK but only occasional checks (from the recording chart kept) on the maintenance of the operating conditions.

On the basis of the above considerations one can set up a model scheme of technical control at metallurgical works (see table).

CALENDAR-SCHEDULE FOR 1959

The caldendar-schedule has been sent us by P. P. Solomenin, senior work-study engineer at the mold preparation shop of the Cheliabinsk Metallurgical Works. It is drawn up on the assumption that on January 1, 1959, team No. 1 begins work on the evening shift. In view of the fact that not all metallurgical plants changed to a 7-hr working day at the same time, for some of them the schedule has to be modified: in the first columns the shift duties should be entered as they occur on December 31, 1958.

Notation:

E – evening shift; D – day (morning) shift; N – night shift; O – day off.

Team 1	E	E	E	E	O	O	N	N	N	N	D	D	D	D	D	O
Team 2	D	D	D	O	E	E	E	E	O	O	N	N	N	N	O	D
Team 3	N	N	O	D	D	D	D	O	E	E	E	E	O	O	N	N
Team 4	O	D	N	N	N	N	O	D	D	D	D	O	E	E	E	E
January	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	
February																1
	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
	18	19	20	21	22	23	24	25	26	27	28					
March												1	2	3	4	5
	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
	22	23	24	25	26	27	28	29	30	31						
April											1	2	3	4	5	6
	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22
	23	24	25	26	27	28	29	30								
May									1	2	3	4	5	6	7	8
	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
	25	26	27	28	29	30	31									
June								1	2	3	4	5	6	7	8	9
	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
	26	27	28	29	30											
July						1	2	3	4	5	6	7	8	9	10	11
	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
	28	29	30	31												
August					1	2	3	4	5	6	7	8	9	10	11	12
	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28
	29	30	31													
September				1	2	3	4	5	6	7	8	9	10	11	12	13
	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
	30															
October		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
November	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
	17	18	19	20	21	22	23	24	25	26	27	28	29	30		
December															1	2
	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
	19	20	21	22	23	24	25	26	27	28	29	30	31			

FROM THE HISTORY OF TECHNOLOGY

THE PAST OF THE IRON AND STEEL INDUSTRY OF THE NORTHWEST

During the excavations of burial mounds in Peterburg* province in 1872-1885, L. K. Ivanovskii found a multitude of metallic objects – knives, axes, sickles, etc. This indicates that even before the XIth century the Northwest of our country had a developed metalwork craft. The craftsmanship of the iron objects found during the excavations is astonishing.

In the Cherepovets district, which is now a large metallurgical center, iron has been mined and worked since olden days. During the excavations at the Troitsk monastery near the town of Beloozero (100 km from Cherepovets) battle axes, knives, hauberks and other objects, which had been made between the Xth and the XIIth centuries, were found. From the XIVth century onward, the Northeast was one of the biggest metallurgical districts at that time.

Iron-producing centers in the Moscow State of XIVth-XVIth centuries were Ustiuzhna-Zheleznopol'skaia in one of the Novgorod districts, the ancient Russian town of Karela (now Priozersk, Leningrad province), Kargopol on the river Onega (now a district center of the Arkhangelsk province), and others. Thus, toward the end of the XVIth century, 119 craftsmen were engaged on working the local metal in Ustiuzhna. In 1633, the government ordered 338,500 cannon-balls (1 to 70 funt each, 25,744 puds total weight)** to be made in Ustiuzhna.

The Northwest supplied the Russian forces with cannons and balls throughout the whole XVth century.

Judging from the taxation, at the end of the XVth century, near the Ilmen Lake alone, over 700 local peasants ("diggers") who were "digging" iron ore, were charged with the "quit-rent" payable to the Novgorod governor; in 1500, on the coast of Finland, hundreds of peasants' bloomeries ("domnitsa") – above-ground furnaces with hand-operated bellows – were in operation. According to academician S. G. Strumilin, at the end of the XVth century, in the districts of Iam, Kopor'e and Oreshki not less than 213 bloomeries with 229 furnaces were recorded, i.e., there was one bloomery ("domnitsa") or forge*** for every 8 households. At the end of the XVth century, iron was produced in the Northwest not only for local needs but also for export.

Conditions for the development of iron manufacture in bloomeries were very favorable in the Northwest region. Bog iron ores lay here at very small depth. Of course, charcoal was used as fuel; there was plenty of wood in the Northwest.

Iron production was not a continuous process at that time: when he had a few days free from agricultural work, the "bloomery-worker" or "smith" would engage in iron making and when necessary would just as easily discontinue it.

Bloomery iron was also produced in other districts of the Northwest, in particular in the Olonets territory, in the Trans-Onega region and in Karelia. Thus, Karelian iron was used for making a number of "tiufiaki" (mattresses) – a peculiar cannon with a tapering bore 140-200 mm in diameter at the mouth. The "tiufiaki" were designed for "shooting stones and other missiles" and, apparently, answered their purpose by covering a large area with shot.

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* Now Leningrad.

** funt (pound) = 0.9028 US lb = 409.512 g; 1 pud = 40 funts = 36.113 US lb.

*** A room in which there were furnaces for producing iron was generally called "domnitsa." Such a room could contain two or even more furnaces.

In spite of their relatively high output, the good quality of the metal and low production costs, the blast-furnace works which appeared near Tula in the first half of the XVIIth century did not result in the elimination of peasants' bloomeries. Iron and iron objects from the Trans-Onega region, the Olonets and Belozersk districts and other parts of the Northwest were sold in Tikhvin from where it was sent to Novgorod, Pskov, Yaroslavl', Moskow and even to Kazan' and Astrakhan. The production of bloomery iron in the Northwest did not cease even when the Urals ferrous industry was established.

In the XVIIth century, however, the demand for metal became so great that Russia had to import metal from abroad at very high prices; in 1692, for instance, when several works around Tula were already in operation, the price of one pud of Swedish rod iron was 5 rubles 50 kopecks in gold (at the rate of exchange for the ruble at the beginning of the XXth century).

An extensive construction of iron works began in the first quarter of the XVIIIth century. From 1700 to 1725, over 200 industrial establishments were set up in Russia. Peterburg became a large industrial center. Ship building was especially well developed here. Out of 900 warships built in Russia between 1700 and 1725, 262 were built in Peterburg.

In 1711, on the bank of the River Neva, where Foundry Avenue now begins, the Foundry Yard was built, and initially it produced copper cannons. In 1721-1723, the state Sestroretsk Works was erected and produced guns, bayonets, pistols, daggers, swords, anchors, nails, sheet iron and wire. Skilled workers for foundry and armory work were brought from Moscow and Tula and from Olonetsk Works.

In 1719, at the Izhora Sawmill the production of galley anchors and copper plates for facing the underwater part of ships was started; the production of anchors, however, did not come under way and was transferred to the Sestroretsk Works. Only in 1753, at the Admiralty Sawmill on the Izhora, the construction of a new anchor work and a copper-processing works was started and these were the forerunners of the present Izhora Works.

Under those conditions, the increased demand for ferrous metals could not be met either by the works near Tula and in the Trans-Onega region or by the small bloomeries scattered over many districts of the country.

The development of Peterburg gave inducement for special attention to be paid to the development of the metallurgical industry in the Northwest. State metallurgical works which, at that time, were considered large, such as Petrovskii, Konchezerskii, Povenetskii and others, were built.

As a result of the successful development of the Ural's metallurgical industry at the beginning of the XVIIth century, the metallurgical works of the Northeast became uneconomic; in addition, the blast furnaces of the beginning of the XVIIth century required much more raw material and fuel than bloomeries, and in the twenties of the XVIIIth century the supplies of ores and wood at Olonetsk works were already markedly depleted. The works, however, were not shut down since they were not far from Peterburg and would be essential in the case of war with Sweden. The Olonetsk works were producing cannons and cannon-balls, guns, swords and bayonets.

There, in 1772, a cast-iron rectangular-bore cannon, 471 kg in weight and with 156 cm long barrel was made. It was charged with three balls at a time on a bottom plate. Specialists for the mining and metallurgical industries were trained in Olonets, where there was a school for training mining engineers.

Later on, the Olonets metallurgical industry, although not developing as rapidly as the Ural industry, was producing a fairly large amount of metal: for instance, in 1800 the Konchezerskii and the Aleksandrovskii Works produced 272,100 puds of pig iron and that constituted about one seventh of metal output of the works in Central Russia. It is interesting to note that these Olonetsk Works were the only two state works out of 42 in Central Russia.

In the second half of the XVIIIth century, just as in the first half, shipbuilding, metal working and munitions production played the leading part among the industries of Peterburg.

In 1765, the Izhora metal-processing works, which belonged to the Admiralty, changed over to the production of large forged anchors. In 1780-1790, an "upsetting machine," for making copper and iron sheets, was installed, the foundry shop and furnaces for melting iron scrap were built and a tool shop and a lathe shop were set up.

Berd's first metal-processing works, producing household goods and later on equipment for industry, came into being in Peterburg in 1792; in the first quarter of the XIXth century, the works produced 130 steam engines and 11 marine steam engines.

At the end of the XVIIIthand the beginning of the XIXth centuries, in Peterburg and its vicinity, state works began to be built once again. In 1797, the Artillery Department set up a new armory shop at the Izhora Works. During the Russian-Turkish War (1787-1791) and the Russian-Swedish war (1788-1790), the erection of the Kronshtadt iron works began and there unfit cannons, shells and other metal products which accumulated in large quantities, were remelted. As early as December 1789, the works was producing 3-, 6- and 12-inch cannon balls, 3-inch grenades and 3-pud incendiary shells. By 1790, the works had also begun to cast guns.

In connection with the danger of a war with England in the beginning of the XIXth century, the works was transferred at the end of March, 1801, to Peterburg, behind the Narva outpost. On the basis of this works, the Peterburg State Iron Works was built, later it was leased to Ogarev, and after 1868 was known as the Putilovskii Works. However, the Kronshtadt Works, which constituted the basis for the Putilovskii Works, continued to operate. Its output of shells for 1808-1815 constituted 90% by value of the total output.

In 1825, on the left bank of the River Neva 7 versts * from the town, was built the Aleksandrovskii Works (now the Proletarskii Works) which later became the property of the Peterburg-Moscow Railroad. It was built by Russian specialists and it was provided mainly with Russian-made equipment.

In the fifties of the XIXth century, Peterburg was the most important center of the metal-working industry in the country: in Russia, in 1859, there were 38 foundries and mechanical-engineering works which employed 4,083 workers and produced goods valued at 2.52 million rubles; 10 works in Peterburg employed 3,094 workers and were producing 1.87 million rubles worth of goods.

Before the reform of 1861, Lessner's Iron Works (founded 1852), the Metal Works (1857) and the Baltic Shipbuilding Works (1856) were built. At that time two mechanical-engineering works, the Siemens-Halske (now the "Elektrosila" Works) and the San-Galli Works, were founded. In 1855, the Neva Shipbuilding and Mechanical-Engineering Works was erected, and, in 1865, the Obukhov Steel Works (now the "Bolshevik Works") was founded; the latter was built under the supervision of Major-General P. M. Obukhov, who organized the production of steel guns in Russia for the first time. For many years, the Obukhov Works was a school for Russian steel makers. D. K. Chernov, A. A. Rzheshotarskii, A. S. Lavrov, N. V. Kalakutskii and many other metallurgists worked there. Initially, crucible steel was made at the Works. Later on, Bessemer converters were installed.

The more developed the machine industry and the metal-working industry in Peterburg, the more acute became the demand for metal.

Russian metal – pig iron and wrought iron – was supplied mainly by the state Olonets Works and by the Gora Blagodat Works in the Urals. After 1830, the private works of General Kandyba in the Peterburg district began to supply metal to Peterburg. The interesting point about this works is that here, apparently for the first time in Russia, hot blast was employed at the blast furnace in May, 1831, and pig iron was smelted partly with the use of wood and peat. Here, in 1829-1840, attempts were made to use Borovichi brown coal for the blast-furnace process.

The establishment and rapid development of the metallurgical industry of the South did not improve the metal supply for Peterburg's machine industry, because Peterburg was situated very far from that metallurgical region, and, in connection with the industrial crisis at the beginning of the XXth century, it was more economical in the Northwest to use imported iron and fuel. The availability of large amounts of iron scrap in Peterburg made it economic to develop a small-scale ferrous industry i.e., iron-smelting shops within machine-building works.

The open-hearth furnaces in Peterburg produced special steel: for guns, armor, tools, etc. The first armor-plate mill in Russia was introduced at the Putilovo Works at the end of the XIXth century. The production of guns – field guns, fort guns and naval guns – was organized at the Putilovo Works and the Obukhov

* verst = 1066.8 m.

Works. Metallurgists, D. K. Chernov and P. M. Obukhov and artillery-men, A. V. Gadolin and N. V. Maievskii, worked in Peterburg. Following D. K. Chernov, the professors of the Peterburg Polytechnical Institute, A. A. Baikov and M. A. Pavlov, founded scientific schools which had a fruitful influence on the development of the home – and world – metallurgical industry.

B. D. Sysoev
Siberia Branch of the Acad. Sci., USSR

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DENDRITIC CRYSTALLIZATION

by **D. D. SARATOVKIN**

2nd Edition,
Revised and Enlarged

Translated from Russian

THIS SIGNIFICANT volume has been extensively revised by the author from the 1953 edition; in particular with *fresh material derived from observations under the stereoscopic microscope.*

The first section deals briefly with some general concepts on crystallization, drawing an important distinction between *genetic* and *structural* types of crystals, including some aspects of the *defect crystal state.* The second section covers at length the illuminating ideas and observations of the 19th-century Russian metallurgist D. K. Chernov, who proposed many of the basic ideas of dendritic crystallization. The third section is an extended survey of current views on dendritic crystallization, in which the ideas of many Soviet and other scientists are briefly summarized and criticized. Section four presents the growth forms of real crystals; all types are reviewed, but only dendritic or closely related forms are selected for subsequent investigation.

Following sections discuss the causes and forms of crystal growth, with *detailed applications* to certain substances that have been extensively studied *(particularly the ammonium halides),* and to eutectics in metal and organic systems; an extensively revised presentation on steel castings which provides a lucid explanation of how the various structures found in real castings can be fitted into the author's theory of dendritic crystallization. Nearly all the concepts developed earlier in the book are utilized in this final section.

The main bulk of the volume contains many *original* and *unpublished* ideas and observations, and is an excellent example of the modern macroscopic approach to the crystalline state by an experienced worker concerned with the infinite variety of real crystals—all of which is enhanced by a *profusion of explanatory line diagrams and sets of stereoscopic photographs.*

CB translations are by bilingual scientists, and include all photographic, diagrammatic and tabular material integral with the text.

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